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March, 1932

A McGraw-Hill Publication

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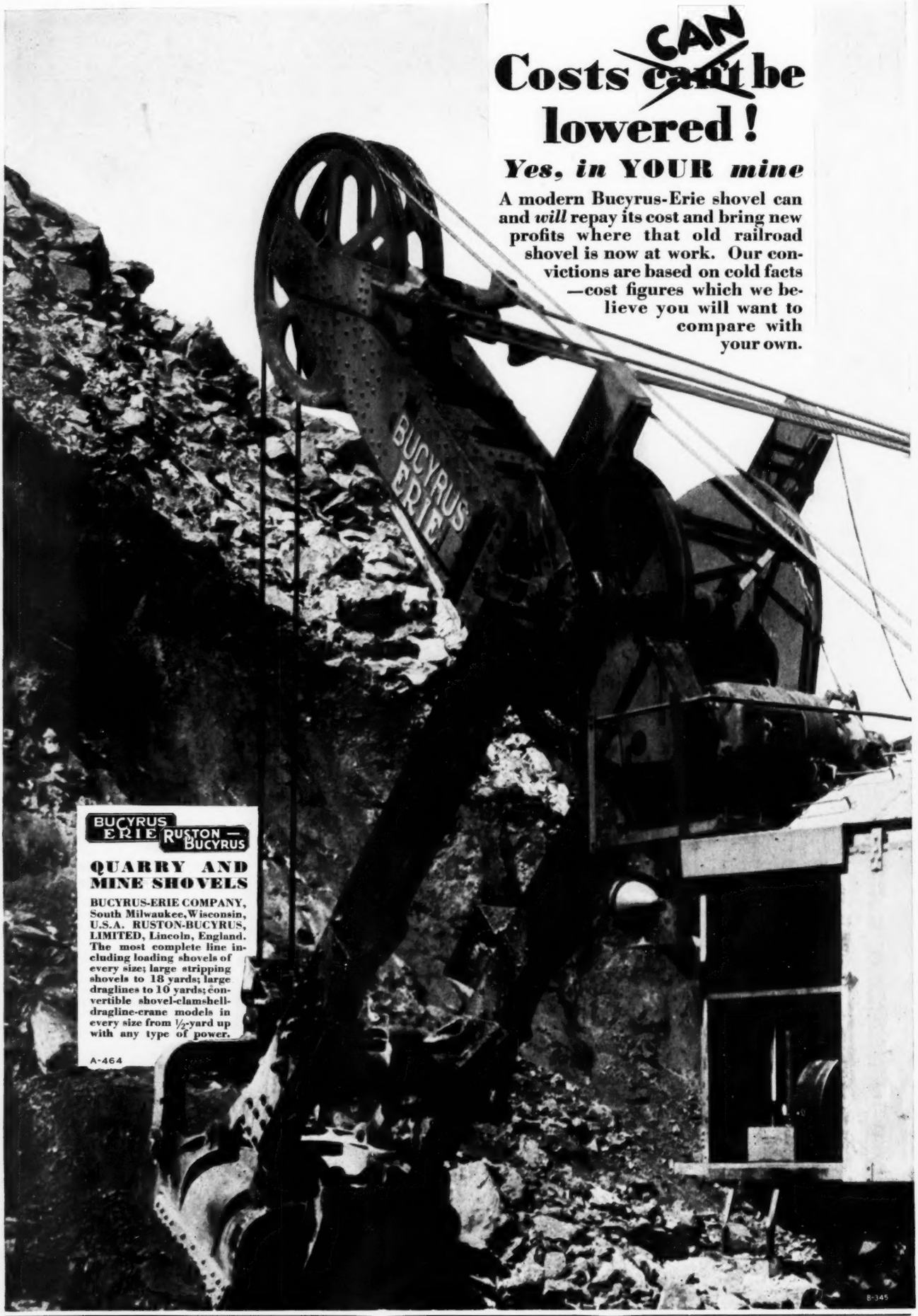
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Engineering and Mining Journal

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Engineering & Mining World

A. W. ALLEN, *Editor*

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330 West 42d Street
New York, U. S. A.
Cable Address: "MCGRAWHILL, N. Y."

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Aldwych House, Aldwych
London, W.C. 2, England
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Engineering & Mining Journal records mine, quarry and plant practice and developments in all the countries of the world. It is published monthly, at New York, U. S. A.

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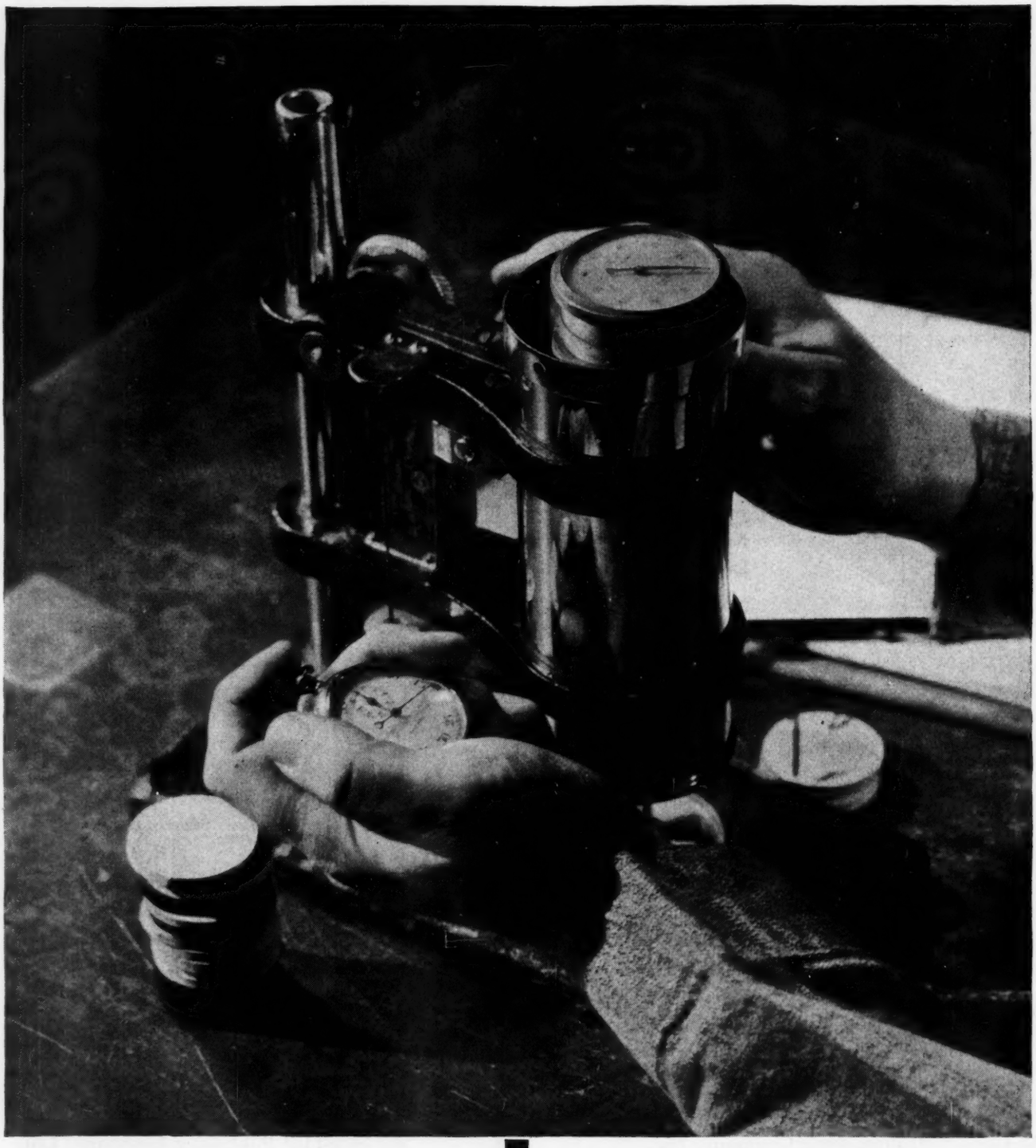

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March, 1932 Vol. 133—No. 3

A. W. ALLEN, *Editor*

Shenandoah-Dives
Meets Changed Conditions

ADAPTABILITY is an important factor in the attainment of success. The man who can change his mode of living to suit altered environment, or the business enterprise that can revamp its structure to meet unexpected and adverse economic conditions, is most likely to succeed. A striking illustration of the validity of this truism is to be found in the report of the Shenandoah-Dives Mining Company for 1931.

In August, 1929, *Engineering and Mining Journal* published an article by Mr. Charles A. Chase, manager of Shenandoah-Dives, describing the successful development of a group of mines in the San Juan basin of Colorado. At that time the company estimated the average value of the ores at about $12 to $13 a ton, of which about half was in gold and most of the remainder in copper and silver, with a small amount in lead. Net smelter returns, per ton of ore mined, were expected to be about $9 per ton; and on that basis the company was building a concentrator with a daily capacity of 300 tons, capable of being enlarged later to treat 600 tons. The plant was put into operation in February, 1930.

Two years have elapsed, and the course of base-metal prices has been steadily downward. Ores that contained $7 in copper, lead, and silver in 1929 contain less than half that value in those metals today. But the drop in metal prices was not the only adverse circumstance that Shenandoah-Dives had to meet. Large-scale operation showed that the metal content of the ore was considerably lower than had been indicated by sampling. In 1931, the actual gross value of the ore mined, on the basis of current metal prices, was $6 per ton (contrasted with the original estimate of $12 to $13); and the net smelter return per ton of ore mined was $4, instead of $9. The company had to operate a new mine on less than half the revenue per ton that had been estimated when it was prepared for production.

Shenandoah-Dives is situated in a region of intense winter cold, and the management must be prepared during that season to operate under conditions involving isolation from the outside world. The oreshoots are narrow. Cost of development is comparatively high; in 1931 it averaged 55 cents per ton of ore mined. Treatment effects the recovery of four metals in a single concentrate. Under these conditions, few would expect a high margin of profit on ore that returns only $9 per ton. But when that ore becomes worth only $4 per ton, operation without financial loss is a difficult task indeed. Yet Shenandoah-Dives made an operating profit of 70 cents per ton on ore mined in 1931. The total operating cost of $3.30 per ton includes development, mining, transportation, milling, and general expenditures. It is 90 cents per ton below the 1930 record!

Mr. Chase's report indicates that the most important factors in this reduction of expenditure were careful supervision of mining methods and the handling of an increased tonnage. By expanded development work, more working faces were made available for production, as a result of which the mine handled a record total of 22,000 tons of ore in December. The mill, rated at a capacity of 300 tons per 24 hours, has averaged well over 550 tons daily without important additions to equipment. Maintenance work is done in anticipation of possible breakdowns, thus avoiding heavy repair charges. Every item of expense is carefully scrutinized with a view to the introduction of further economies.

The aggregate value of the silver, lead, and copper in the ores when Shenandoah-Dives started production was as great as the value of the gold. Today, Shenandoah-Dives is a gold mine with a byproduct recovery of the three other metals. More than two-thirds of its income is obtained from gold, and on that basis the maintenance of production and profits seems assured. Not the smallest factor that presages success, however, will be continued good management.

Oxygen-Free Copper
Reaches Production Stage

THE furnace refining of copper, although an empirical process, requires a high degree of skill. Little fundamental progress in technique has been made since its inception. In comparison with the other branches of copper metallurgy, the final furnace treatment appears lengthy and crude. To a great extent it is concerned with the removal of impurities, dissolved or free, not derived from the cathodes but resulting from contaminants acquired by the metal during the process. The relation of these impurities to each other, their influence on the physical qualities of the copper and on the "set," an important guide in casting operations—these are matters of the greatest importance to the refiner.

An interesting advance in the art has recently been accomplished at the Carteret plant of the United States Metal Refining Company, where extensive research has led to the commercial production of standard shapes of refined copper with an oxygen content so low that it cannot be detected by the customary microscopic method. This is accomplished by casting the copper in vertical water-cooled molds, thereby protecting the metal from oxidation during pouring, solidification, and cooling.

Precise control of the temperature of melting, exact regulation of pouring speed, and other essential refinements in operation technique insure a metal uniform in quality and free from defects.

The achievement of production, on a commercial scale, of copper that is free from oxygen will be commended by a group of metallurgists who affirm that a material with such qualities possesses for certain purposes distinct advantages over the regular tough-pitch copper, a slightly higher cost of production being the main obstacle to widespread use. The claim that oxygen is necessary to counteract the influence of impurities such as iron, lead, tin, or sulphur, although backed by accumulated experience, should lose strength as the metallurgist learns to keep such impurities out of the metal. The electrolytically deposited copper of today is generally of sufficient purity to make such protection superfluous. The fact must be admitted that oxygen-free copper possesses distinct advantages, such as greater ductility and density, a somewhat higher electrical conductivity, and higher fatigue resistance. It may be used under a condition involving contact with reducing gases at elevated temperatures, which causes embrittlement of regular copper.

One reason for the difficulty of studying the effect of oxygen and other gases in copper is that small-scale laboratory tests give results difficult and sometimes impossible to interpret. This partly accounts for the fact that changes made in the process have been few. The "set" and the physical properties of the copper are influenced by factors such as mold temperature, rate of cooling, depth of the metal in the mold, area of metal exposed during solidification, height from which the molten metal falls during pouring, method of filling the mold, and many others impossible to duplicate on a small scale. With commercial shapes available that are cast from a full-size furnace, a more comprehensive study should be possible. Such a study should disclose valuable facts that might well permit the entrance of the metal into entirely new fields. Six years ago Professor C. R. Hayward, of Massachusetts Institute of Technology, urged copper metallurgists not to rest satisfied until they were able to produce a copper of maximum purity, predicting that such a metal would be found to possess properties in many respects superior to the best commercial grades of the day.

Wage Scale Not the Only Labor-Cost Factor

A FACTOR that every consultant must consider in reporting on a mining prospect is the question of local labor conditions. As the mining engineer makes the whole world his terrain, the variations he encounters are almost infinite. Wage scales range from a few cents a day paid in the Congo, in Malaya, or in New Guinea, up to more than $8 a day—the average earnings of contract labor at the Alaska Juneau mine, in Alaska. Wide divergences such as these are startling. They may be turned to use by the American politician who expresses concern about standards of living, and, in fact, they frequently have been so used.

Sometimes, however, the agitation may be carried too far. In the recent campaign in the United States for a tariff on copper, some of its advocates aver that the American miner is forced to compete with "slave labor" in Africa or "peon labor" in South America. These charges, particularly when they are made by a United States Senator, do not meet with a kindly reaction abroad. The Brussels correspondent of *Engineering and Mining Journal* reported recently that the Belgian Government has protested to the United States about allegations made in Washington that slave labor is employed in the Katanga mines. American companies operating in South America, where they have built modern communities in isolated regions, are unable to make protest through diplomatic channels, but they also feel maligned.

Fortunately, the report on copper of the United States Tariff Commission, recently submitted to the Senate, throws light on this disputed point. It states that mine-labor cost in Africa, per ton of copper ore produced, is $1.75. In the United States it is about 98 cents. Because the African ore is of a higher grade, the mine-labor cost per pound of copper is 1.61 cents there, 3.08 cents per pound in the United States. In regard to South America, the commission states that "the service performed for equivalent remuneration was in the ratio of 100 in the United States to about 88 in South America." Throughout the entire report, which dealt largely with copper-production costs, one can find evidence that the advantage of the foreign mines lies in most cases in the higher copper content of the ores, not in low labor costs.

Looking Backward or Forward in Plant Operation

MECHANIZATION of industry and the machine age in general have had their critics on several counts. The evils attributed to them have ranged from nervous strain to loss of craftsmanship and, latterly, to technological unemployment. To these charges the critics have usually added blame for the dirt and ugliness characteristic of so many plants and factory towns. They now find new ammunition in certain elements of the situation wrought in places by the unprecedented economic conditions prevailing. "The old methods are the best," they say, pointing now to an occasional plant where hand labor has been re-adopted in whole or part and the machine supplanted; now to a mill where the owners refuse to modernize on the ground that dividends are more important and can be paid with obsolete equipment. To bolster their position, the critics echo the objections of the plant executive who contends that labor-saving machinery is expensive, that power to run it must be bought or made, that lack of skilled labor will cause high maintenance costs, and that hand methods are more flexible—that the number of employees can easily be changed to fit conditions.

Seemingly these plant owners and executives have been blinded by the exigencies of the unemployment situation. Their attitude is largely the result of the pressure exerted upon them by public opinion and a national emergency—to keep men employed in the face of a greatly decreased demand for their product. Circumstances can readily be conceived where only by some such conversion of the mode and scale of working can they meet this dilemma and provide maximum employment at a minimum loss.

Obviously, this applies only where the nature of the operation is such that the work previously done by machine can be performed manually without other equipment, as, for example, shoveling in an open pit.

The objections cited are easily answered. Labor-saving machinery and power supply may be "expensive," but capital investment can be returned by lowered costs, if the equipment be properly selected and the life of the project sufficiently long. Under these conditions, moreover, time will permit the training of labor to enable it to handle the equipment with minimum or reasonable maintenance. Exceptions occur: demand for the product may normally so fluctuate that labor-saving machinery cannot be used to advantage, especially if at the same time labor be abundant. Everything considered, the points of the argument cited fail to impress. With the world's present economic troubles eventually overcome, modernization of plant may be expected to continue. Competition, if nothing else, will force it. The wise operator whose equipment needs overhauling and replacing will not ignore the opportunities of lowered prices and abundant labor.

Available Information And Technical Success

ALMOST every mail brings to the editorial department of this publication further evidence in support of a recommendation made in a recent issue: Keep your back copies of *Engineering and Mining Journal*, for you will surely need them; or at least clip and classify articles in those subjects in which you are primarily interested. An efficient system is necessary, of course; and a well-known engineer, who finds a card index useful, contributes the following helpful suggestions:

> "A brief index, under the proper headings, of such articles as one feels one may find useful takes only a few minutes each week. An additional line or two indicating the scope sometimes helps, and saves digging out an article having no application to the problem on hand. I find this particularly true with the earlier of the 80 volumes I have. Even when extended to cover many periodicals, such an index rarely requires half an hour a week, though it is better written while the matter is fresh in the mind. Abbreviations must be many, and both sides of the card used, or it becomes too bulky. A typewriter that will write close to the top and bottom and ends of the cards is needed. Dictation produces more bulky results than personal indexing. My 2,000 cards, which I estimate have 20,000 to 25,000 references, fill two metal boxes, each 5x10½x3½ in., with a third well under way."
>
> "Fortified with such as these, an engineer may sojourn for months where only airplanes go, and send for whatever copies of *Engineering and Mining Journal* he needs to help solve each problem as it arises."

For the itinerant mining engineer, such a method has the merit of simplicity and economy and efficiency. Those whose work is more centralized, however, may prefer to bind the volume of issues with the index—distributed with the last number of each year. Such binding preserves all pages and all issues intact, and is therefore favored as a protection against careless borrowers. A purchased binder may be used, in which the volume of issues is held by means of flexible brass prongs that are passed through holes punched in the margin. Or, if such a binder be not available, a workmanlike job of binding can be done, at negligible cost, as follows:

> Covers, wire staples, and all matter except text are first removed. The pack of issues that constitutes the volume is then stacked in order, the index placed first. Two pieces of two-inch board are then taken, one 14x3 inches and the other 16x3 inches. The shorter is placed underneath the volume, in line with and about one-half inch away from the binding edge, extending one inch at each end beyond the top and bottom edges. The second piece is then placed in the same position on top of the volume, to extend one inch at one end and three inches at the other. These binding boards are then firmly held in place by clamps or bolts. With the longer board toward the operator, and its extension to the right, the pack is then placed so that the lower edge of the shorter board rests on the edge of a bench, the extension of the longer piece being held in a vise. The binding edge is then sawed to a depth of one-quarter or three-eighths of an inch, about five notches being made, spaced equidistant. The backbone is then well painted with white glue, which has been liquefied by heat. A piece of strong cord is then taken; and, starting at an end notch, this is laid in the notches, from one side to the other, until the last is reached; then passed back in the same manner and tied to the starting end. Another coat of glue is added, to be covered with a strip of strong muslin. The glue must be allowed to set before the clamps are removed. A covering of stout brown paper or other suitable material may be used as a protection of the first and last pages of the volume.

Many other systems to facilitate reference and to prevent loss of issues or articles have been evolved. If *Engineering and Mining Journal* is worth buying, it is worth saving. Keep your back copies of the magazine, for you will surely need them.

Internal vs. External Force in Rock Crushing

THE heating of an ore before crushing renders it more friable. Sufficient moisture has usually penetrated the material when *in situ* to cause the formation of initial fracture, if not actual rupture. This rupturing is well marked in oxidizable ores, into the interior of which water, carrying oxygen, follows lines or planes of cleavage developed at the surface.

Messrs. R. S. Dean and John Gross, of the United States Bureau of Mines, recently concluded preliminary experiments, described in a Report of Investigations, on the effect of controlling pressure during the heating stage of an attempt to demonstrate the practicability of decrepitation, as a substitute for crushing, so that steam will be formed on release of that pressure. Moisture in the specimens of rock was not determined, apparently, but a preliminary soaking in water for several days was reported as necessary. Pressures up to 180 pounds per square inch were generated in a test cylinder; and the results are claimed to be sufficiently encouraging to justify a well-outlined program of research and experimentation. From the standpoint of cost, the proposal to substitute for crushing by impact and abrasion a shattering by the explosive power of steam, by heating the moistened ore under pressure and then releasing the pressure, is reported by the chief engineer of the metallurgical division of the Bureau and his associate, authors of the paper, to presage great possibilities.

Recent Developments in Underground and Surface Practice

Synopses of selected papers and comments on ten subjects of outstanding interest discussed at the annual meeting of the A.I.M.E., held at New York, February 15-18, 1932

In terms of number of ores, non-sulphide flotation is more important than sulphide flotation, state Messrs. Coghill and Clemmer, supervising engineer and junior metallurgist, respectively, at the Mississippi Valley Station of the U. S. Bureau of Mines. Fourteen "ores" have been investigated; namely, limestone, phosphate rock, bauxite, fluorspar, rhodochrosite, manganese oxides, barite, siderite, chromite, sheelite, ferberite, cyanite, beryl, and spodumene. Of these, four have been put through successful trials in commercial flotation plants; namely, phosphate rock, fluorspar, rhodochrosite, and manganese oxides. Laboratory tests indicate that some of the other ores mentioned are equally amenable. . . . The flotation of such ores depends on the proper selection and amounts of pine oil, sodium oleate, oleic acid, sodium carbonate, sodium hydrate, sodium silicate, and similar reagents. Repeated tests have invariably shown that some of the ores require a little sulphuric acid in the cleaning treatment. Generally less than one hundred parts of any one of the reagents in one million parts of water is sufficient. A slight excess is likely to destroy effective operation of flotation action.

Will H. Coghill

J. Bruce Clemmer

Soap Flotation of the Non-sulphides

The flotation process has been so closely allied with many of the sulphide minerals and their early and associated oily reagents that the term "oil flotation" has erroneously been applied to the entire flotation process. Today, the term "chemical flotation" might be used to advantage. Flotation literature has dealt primarily with the removal of sulphides from non-sulphide gangues, and the separation of one sulphide from another. Now, the application of the process to the separation of non-sulphide minerals is coming to the fore; and from the standpoint of the number of commercial minerals involved, the flotation of non-sulphides offers the larger field. Here, soap or soap-forming substances, such as the fatty acids and their derivatives, are used; hence there may well be a subdivision of the chemical flotation process called "soap flotation."

Early experimenters learned that it was easy to form a froth or foam that would carry a part or all of the ore charge over the lip of the flotation machine; but this could not be regarded as flotation, because of the lack of selectivity. Doubtless soap was one of the first reagents tried, and doubtless it was rejected on account of its gangue-carrying properties. Now, this same gangue-carrying ability makes soap an acceptable reagent, for we are dealing with that which, among sulphides, is considered "gangue." As in the flotation of sulphides, so in the flotation of non-sulphides, the selective flocculation of the mineral and the dispersion of the gangue are the first objectives. In fact, these two phenomena epitomize flotation. They depend upon the unlike surface properties of the respective minerals; and as long as the components of an ore have unlike surface properties, a separation is possible, at least in theory.

Often, in non-sulphide flotation, it makes little difference which mineral floats and which is depressed; generally only a separation is sought. Also, for economic success, the separation does not always have to come up to the standards demanded in sulphide flotation. When two or more minerals in the ore are equally floatable, or when the minerals themselves are not pure, clean concentrates cannot be obtained. These conditions prevail more often in non-sulphide than in sulphide ores. However, some of the non-sulphide ores which have clean mineral grains have yielded soap-flotation concentrates as clean as the best sulphide concentrates. The metallurgical limitations are probably more inherent in the ore than in the soap flotation process.

Although, as is commonly known, soap makes a suds, its rôle in the flotation discussed here seems to be due to its selective absorption. Selective adsorption implies the use of a bare trace of reagent, and in soap flotation the amount of soap rarely exceeds 100 to 150 parts per million parts of water. This small amount is generally enough to flocculate both the mineral and, to

some extent, the gangue. This latter effect must be minimized by using alkaline dispersing agents like sodium silicate or sodium carbonate. Seldom is soap sufficiently selective to be used alone. Pine oil generally is necessary to impart brittleness to the froth. Consequently, the term "soap flotation" is not strictly technical, because the froths are regarded as modified pine-oil froths.

Sodium silicate is an important reagent in non-sulphide flotation, but unfortunately the term is not specific. The ratio of soda to silica is seldom considered. Many of the silicates were examined, and, although any number of the series may be used, the orthosilicates and meta-silicates were preferred. Bicarbonates are undesirable as alkaline dispersants. In fact, suspicion has rested on the carbon dioxide naturally contained in the air passed into the pulp. To determine the effect of carbon dioxide, comparative flotation runs were made with natural and with "scrubbed" air. The latter, where the carbon dioxide had been removed by passing the air through a solution of caustic, gave the better results. If this observation is correct, pneumatic machines that employ contaminated air as a means of agitation should be avoided in soap flotation.

Fatty acid reagents are changed instantly upon entering hard water. Very few non-sulphide ores are without the accessory soluble salts that make water hard. Some hardness is allowable, but the excessive amount of hardness induced by non-sulphide ores from the arid Southwest has hindered good flotation. In the study of the effect of hard water, calcium oleate was prepared and used as a flotation agent. The flotation was satisfactory. An elevated temperature seems to make hard water more tolerable.

The non-sulphide minerals which possess distinct crystalline structure and luster are the most floatable. Earthy minerals are generally difficult to concentrate. Wad manganese ores are especially difficult.

Strangely, sulphuric acid has been a valuable reagent in some of the soap flotation of non-sulphides. Sometimes it is used in the cleaning of the rougher concentrate; but when the ore is free from gangue slime, it may be used advantageously in the rougher. When coarse gangue contaminates the cleaner concentrate, a trace of dilute sulphuric acid will depress it.

Mechanical machines with a positive means for controlling the air are best. In some of these the leak of air through the stuffing box which constitutes a part of the aeration unit is objectionable because it prevents regulation of air through the valve. Mechanical beating in the presence of a moderate feed of air is best in soap flotation.

Sand Filling Through Pipes

Choice of the proper method to be used in placing sand fill, states Mr. Eaton, internationally known consulting mining engineer, must depend on local conditions. The hydraulic method, assisted by the injection of compressed air in any long lateral pipe lines, is probably best for relatively shallow mines, where ample pumping capacity is available. . . . In a deep mine, where the pumping charge would be excessive, the pneumatic method would be preferable, especially if coarse material is used. . . . Under the most difficult conditions the best system would seem to be a comparatively thick pulp transported through rubber-lined pipe, with pneumatic boosters at suitable intervals. . . . Selection of the most suitable diameter of pipe depends on the size of the material to be transported and the cost of the pipe. A certain velocity must be maintained, so that the pipe will not clog, but too high a velocity will invariably result in excessive wear.

Lucien Eaton

Of all the materials used for underground filling the best is without question clean sand. Outstanding advantages of sand are its comparatively slight compressibility and the ease, economy, and convenience with which it may be placed. On account of the voids which necessarily occur in fills made of coarse waste, such fills will compress 10 to 25 per cent before they develop their full strength, whereas clean sand seldom compresses more than 5 per cent. Sand is especially desirable when practically complete filling is required, and sand derived from mill tailings, especially when not thoroughly cleaned, has been remarkably effective in sealing off and extinguishing mine fires.

Sand may be transported and placed underground by several methods, which may be classified as follows:

1. *By Hand*—The sand, which must be comparatively dry, is taken down the shaft in cars on cages and is distributed to its destination in these cars, or it is dumped down raises and is drawn off and distributed by means of cars, wheelbarrows, or scrapers.

2. *By Water*—The sand is mixed with water, and is flushed down pipes or boreholes, and is distributed by pipes or launders. The material so distributed may be as low as 20 per cent or as high as 75 per cent solids, depending on the size of the particles and the means of distribution.

3. *Compressed Air* — Dry sand dumped down raises or large pipes, from which it is drawn off and distributed through small pipes. Two systems are in vogue: (*a*) An intermittent high-pressure system, in which the sand is charged periodically into a tank, from which it is blown through pipes to its destination, by compressed air at a pressure of four to five atmospheres. (*b*) A continuous low-pressure system, in which the sand is fed into a pipe through which a large volume of air, at a pressure of about one-half an atmosphere, is flowing.

4. *By a Combination of Compressed Air and Water*—A mixture of sand and water is sent down through pipes or boreholes, and is distributed through pipes, into which small jets of compressed air are introduced at intervals.

The cost of placing fill by hand is usually high, except under conditions where the wages of unskilled labor are very low, and to make a complete fill by the hand method is always difficult.

Hydraulic methods of filling have been used in many parts of the world. One of the chief disadvantages of the hydraulic method is the cost of pumping back to surface the water used for transporting the sand. Another important item of expense is the wear on the pipes used for transporting and distributing the filling material. This wear may be reduced to a small amount, if special pipes are used, which, however, materially increase the first cost.

Compressed air has been used successfully for placing dry sand-fill. Coarse material is handled readily, the diameter of the largest particles being one-third of the diameter of the pipe through which they must pass. The principal disadvantages of the method are the cost of compressing the air and the frictional wear on the pipes, especially at the turns. The low-pressure system costs less for air compression but uses larger pipes.

The combination of compressed air and water is characterized by the same disadvantages as the hydraulic and compressed air systems, but not to as great an extent as either of them. Use of compressed air makes it possible to reduce the percentage of water in the mixture, thereby reducing the pumping cost, and it also makes possible the distribution of material through long lateral pipes, and prevents the settlement of sand in low spots in the line. These advantages more than offset the cost of the compressed air.

At the Hodbarrow mine, at Millom in Cumberland, England, beach sand,

placed by a combination of water and compressed air, is being used as filling. This great mine has been in operation for more than 100 years, and has produced over 22,000,000 tons of ore. The ore is a hydrated hematite and is overlain by sand and other loose material. It is on the seashore, and the area that has been undermined is protected from the ocean by a sea wall. Most of the ore has been mined by top-slicing, but the difficulties brought on by subsidence became so serious that it was desirable to change the system of mining. The process has been reversed, and the ore is now mined from the bottom upward, the slices being filled instead of being caved.

Filling material is sent into the mine through 4-in. pipes with Victaulic joints. The pipes are hung in boreholes or in disused shafts. At the collar of each pipe is a concrete bin or hopper, and over the opening of the pipe is a wire screen about 1 ft. square with 1-in. openings. If the sand is near enough to the borehole, it is flushed into the hopper with the aid of a hose.

Most of the sand used, however, is loaded into cars with a clamshell bucket, and is dumped into the bin, from which it is washed into the pipe by a 2-in. hose with a 1-in. nozzle. The stream normally carries 40 per cent solids and 60 per cent water, but sometimes this percentage is reversed, or the percentage of solids may fall as low as 30 per cent. All of the sand grains are rounded, and, for this reason apparently, the wear on the pipes is not serious.

In the operation of this system the principal feature is the use of compressed air as a booster. The pipe has long-sweep elbows, and into some of these $\frac{3}{4}$-in. pipe is tapped on the center line of the pipe on the exit side, so that the air jet will not impinge upon the wall of the pipe. The $\frac{3}{4}$-in. pipe is reduced to $\frac{1}{8}$-in. diameter at the end. The compressed air increases the velocity of the stream, and keeps the pipe clear of sand without regard to the gradient. This makes removal of sags and low places in the line unnecessary, and the velocity of the discharge helps in filling the upper portions of the stope.

As the free discharge of air at 70 lb. through a $\frac{1}{8}$-in. nozzle is only 19 cu.ft. per minute, air consumption is not a heavy item. Moreover, as the mine is less than 600 ft. deep, the pumping charge for the water from the fill cannot become a large item of expense.

Mining Geology and Mine Exploration

Mining geologists have been charged so often with having failed to make additions to the number of mining districts that further reference to the matter in this paper will probably find them insensitive. The author admits the value of mining geology in an established camp, and concedes that the mining geologist, by his regional mapping and suggestions, has often directed the efforts of prospectors to regions where eventually important finds have been made. But he questions whether or not the mine operator and mining geologist will be able to make the new discoveries necessary to keep up our future supply of minerals. Evidently, he thinks the field geologist at fault in having been too reticent about his practical experience; also that less attention should be paid to theory and more to results of study of phenomena observed in the field. Part of the blame for the lack of new discovery he places also on the mining capitalist of the second generation, who lacks the pioneering spirit of the founder of his fortune. The treatment of this subject afresh by Mr. Hoffmann, who himself is a mining geologist, will prick those who feel they are being criticised, and win echoes of approval from those who share his views. Part of his paper is abstracted briefly in the paragraphs that follow.

R. D. Hoffmann

Many of the deficiencies that are encountered in mining geology lie, apparently, with the mining geologists themselves. In glancing through the literature of ore deposits, one immediately notices a predominance of articles written by university professors and students. Naturally, subjects which can best be handled in a university laboratory are treated—paragenesis of minerals, theories of genesis of ore deposits, secondary enrichment, and the microscopic examination of ores and rocks. So the literature on mine exploration is mainly academic. The fault here lies with the field men, for too few have written the results of their experiences. When one compares the detailed descriptions of oil fields with those of most mining camps, mining geologists suffer greatly in comparison. Because of wide dissemination of knowledge of oil occurrences and close cooperation between field men, the oil industry has had such widespread success in exploration as compared with indifferent results of most mining companies.

Most of us tend to think of ore deposits as being closely bound up with the present surface of the earth. This may be true of certain types of deposits genetically connected with the late Tertiary volcanics, but clearly not with most sulphide replacements and with Pre-Cambrian ores. We speak glibly of this deposit being formed at great depths and at high temperatures, of that one at intermediate depths and at moderate temperatures. Actually, we do not know the surface expressions of such hidden deposits and still less would we dare to sink boldly to depths of 1,000 to 4,000 ft. in search of such orebodies where little mineralization is evidenced on the surface. I. B. Joralemon, in discussing the Pioneer gold mine of British Columbia, states in a recent issue of *Engineering and Mining Journal*, "If the surface were a few hundred feet higher, only narrow dikes of diorite and very irregular, narrow veins with occasional gold-bearing streaks would be exposed. In all probability the pioneer orebodies would then never have been discovered." Here is a great field open for geological research. In the literature of the subject, an approach to the problem has already been made by Augustus Locke and Paul Billingsley.

That exploration flourishes and languishes with the supply of money available for such work and with the actual need of metals is axiomatic. The attitude of officials of mining companies is the controlling factor. As large companies have grown, many of the original founders have died. Often they have been succeeded by boards, variously called executive, financial, and the like, assisted by large technical staffs with little or no financial interest in the organization. Inevitably the companies have tended toward a bureaucratic form of control. Decrease of initiative in the field has been a natural consequence. More than a mere coincidence is the fact that with the passing of these old pioneers the discovery of new mines has decreased. The offspring of these pioneers, too, have done little to con-

tribute to mining exploration. As a rule they have become a "bond generation," living on interest from capital already created; and, having found a comfortable niche for themselves, they avoid the exploratory risks that their fathers and grandfathers before them would have taken.

Control of mining exploration is now centered in the hands of mining engineers, metallurgists, mill men, and a small sprinkling of mining geologists. Relegation by mine executives of mining geologists to a minor rôle in exploration control is more or less justified by their record in the past decade. However, this summary of the worth of mining geologists has been influenced by the work done by university professors and graduates who are hired for summer or occasional field work. These men, who spend nine months each year working for salaries of $2,000 to $10,000, are appalled, generally, at the thought of being responsible for expenditures of $20,000 to $100,000 on prospects which often appear to be of questionable merit. Their character and training cause them to react against speculative development, and the discovery of new mines is hindered and delayed accordingly.

Rising cost of exploration has likewise caused redoubled caution in those in charge of exploration. The cost has risen in almost geometric progression over the past decade. James Wade, manager of the Tintic Standard mine, in describing the Tintic area, has estimated it cost but a few dollars to find the surface mines, $50,000 to $100,000 the mines on the dip, and $500,000 to $1,000,000 the deep-buried mines in the East Tintic area. Exploration, except for a few virgin difficult-of-access areas, is now beyond the simple stage of surface discoveries. A much different approach from the cursory methods of the past is now required. Will the mining geologist, geophysicist, mine operator, or some combination of these men be able to furnish the new technique required.

In the meantime, prior to the inevitable "pick-up" in exploration, mining geologists might well take advantage of the lull to repair and strengthen their fences. The writer of this paper suggests the following: honest consideration of what is fact and what is theory with a realization that each has its proper place; wider publication by field men of their experiences; recognition that regional structures are responsible for localization of camps, as shown by B. S. Butler's work; publication of "case" structural histories of various deposits; recognition of proper balance between field and classroom; recognition of the fact that a new era of mine exploration is at hand, and that, consequently, new surface criteria of hidden ore deposits must be developed.

Prospecting in the Shield Areas

In this paper the director of the Minnesota Geological Survey has made a statistical study of the major gold-producing regions of the shield areas of the earth. Specifically, he deals with those of the Canadian shield, of the north-central Siberian area, of the Kolar district in India, of Southern Rhodesia, and of Western Australia. The facts appear to show that gold-bearing lodes are most likely to be found in certain relationships to batholithic intrusions of granitic rocks, at or near the contact of the latter with the older rocks they have intruded. Why this is so is a matter for theorists. Recognition that it is so, however, should be helpful to those bewildered by geologic theory. Professor Emmons' deductions, added to previous knowledge, should be a useful guide in selecting, in any given region, the most likely places for new discoveries, at the same time making possible the elimination of large areas as unpromising, without as great a risk of error as might attend the lack of such a guide. The old precept, "ore is where ore is," remains undeniable as ever, but a knowledge as to where it usually is should be advantageous to all concerned.

W. H. Emmons

Geology of the shield areas is characterized by the occurrence of large areas of granitic rocks, including granitic gneisses, granodiorites, and other acid intrusives here embraced in the general term "granite." The granites are intruded into schists, ancient lavas, and other rocks here designated as the "invaded" rocks. They are parts of what once were great batholiths, some of which are scores or hundreds of miles across and contain islands or "roof pendants" of the invaded rocks at places where the roofs of the batholiths were low—that is, where erosion has not yet completely unroofed the batholiths.

In these islands of older rocks, and in the narrow marginal belts and cupolas of the batholiths, practically all of the gold lode deposits of the shields are found. Valuable lodes are lacking generally in the granites themselves except in granite within a mile or less from the contacts with rocks of the islands.

Not all places in the islands of older rocks are equally favorable for prospecting. Most of the greatest gold districts of the shields are well within the islands, and the majority of the most productive districts are near small stocklike intrusives of the granites, or of porphyries inclosed in the invaded rocks, or near fingers of the granitic rocks that extend inward from the larger masses into the invaded rocks. The most valuable deposits, moreover, seem to be located around the smaller stocklike intrusives, whereas the larger ones are generally associated with less productive lodes.

One may classify the regions of the shields, according to their positions with respect to granitic intrusives, as follows, the least productive first:

1. Granite areas one mile or more from a contact with invaded rocks—Lodes are numerous, but generally not workable. Few deposits are productive.

2. Granite less than one mile from contact—A few valuable lodes have been worked, but most of them are small and only moderately productive compared with lodes of following groups.

3. Invaded rocks of the "island" areas—These contain numerous deposits, many of which have been profitably worked.

4. Areas of the invaded rocks near small stocklike intrusives—These contain most of the largest gold lode mining districts in the shields.

Most lode ores of metals are formed in and near the roofs of such batholiths by deposition from solutions expressed upward from the batholiths as they cooled. The latter themselves are essentially barren except in their upward bulges, or "cupolas," on the roofs and in the regions near the warped planes of the roofs. A "dead line" may be drawn on a section of a batholith, below which valuable deposits are rarely found. This dead line lies nearer the roof in its lower regions than in its higher ones. It is about three miles or a little more from the roof in the summit areas, but about one mile from the roof in the trough areas. The valuable gold lodes in deeply eroded areas of granitic batholiths are found, therefore, in and near roof pendants on the batholith. Rarely are they found in the batholith more than one mile from the contact with invaded rocks.

Cupolas, or small upward bulges, are the favored positions at all places on the roofs. They may be found anywhere on the roof; but, for purposes of study and comparison, they are grouped in three classes—summit cupolas, intermediate cupolas, and trough cupolas. Gold deposits associated with cupolas that are treated in this paper are associated with trough cupolas and intermediate cupolas.

The cupolas are generally elongated. Some of them contain valuable lodes, but in many the chief lodes are in the invaded rocks near them. In a large number of gold-bearing districts a "finger" of the cupola, or of the invading rock that surrounds the roof pendant, points toward the gold-bearing area. That seems to be true at Porcupine, Kirkland Lake, Shamva, Kolar, Kalgoorlie, and elsewhere; also on the Mother Lode region of California, although its ore was deposited in a relatively high region, compared with deposits of the shields.

In certain great gold-bearing districts no conclusive evidence that the gold lodes lie above cupolas has been adduced. Erosion is not deep enough to expose the cupolas if they exist. Where erosion has reached the stage at which study may be made of the relations of the deposits to the cupolas, the major deposits are usually found to have been formed in and around the latter. One may infer, therefore, that most of the larger deposits of gold of the deep-seated type have formed in and around cupolas, and that these relations would generally be shown if erosion had reached the stage suitable for the study of their relations to the mineralizing masses. Why gold lodes are concentrated in and around and above granitic cupolas is a matter of speculation.

The deposits described in the paper and on the study of which Professor Emmons bases the foregoing generalizations include those of the Canadian shield, among them Porcupine, Kirkland Lake and Rouyn; those of north-central Siberia, as in the upper valleys of the Lena and Yenissei rivers; the Kolar field in India, the Shamva and others in Southern Rhodesia; and Kalgoorlie in Western Australia. The list is impressive, comprising many of the most productive areas of the ancient shields.

Results of Wire-Saw Tests

Realization of the great waste of materials in quarrying and manufacturing slate led to the initial experiments in the United States with the wire saw. Active interest was taken in the device by several Pennsylvania slate companies in 1927, and the year following sufficient tests were made with it under actual operating conditions to assure its successful application at both the so-called soft-vein and hard-vein slate properties. In the next few years the wire saw was generally adopted by slate quarry operators. . . . Inasmuch as the outstanding advantages of the device are in conserving material, in providing smooth surfaces, and in cutting without shattering, its subsequent appeal to producers of dimension stone is readily understood. . . . Among the progressive stone-quarrying companies that soon expressed an interest in the wire saw was the Bloomington Limestone Company, with which company the author is associated as a mining engineer. In the last few years Mr. Newsom has made extensive tests with the wire saw in the company's quarries. Results of these tests, which he describes here, are significant. For instance, they indicate a 50 per cent decrease in stone-cutting costs, a substantial reduction in waste, and a total saving of more than a million dollars to the Indiana limestone industry. Undoubtedly, Mr. Newsom's results will lead to a thorough investigation of the wire saw by operators quarrying the other building stones.

J. B. Newsom

In the course of operations in July and August, 1931, the Bloomington Limestone Company, at Bloomington, Ind., ran a single wire saw on ledge No. 2 at Maple Hill quarry. The ledge was hard rock, much harder than the average Indiana limestone. It was opened at two ends for a depth of about 20 ft., leaving a 130-ft. section between. During the tests 21 cuts were made in this ledge, to an average depth of 10.42 ft., for a total effective cutting area of 28,445 sq.ft. The actual area sawed was about 20 per cent higher, as we did not level down in the center, preferring to cut deeper on the ends. The various cuts were carefully logged and the results plotted. A study of these results is interesting, as it enables a forecast of probable future costs to be made, and also because it points out the limitations of wire saws, and the lines development should logically take. Some of the inferences to be drawn are discussed here.

No trouble was due to length of cut, except that we could not easily level down a long cut, and it is probable that longer cuts, say up to 200 ft., can be made. Obviously, longer cuts will result in a higher sawing rate in square feet per hour if they do not cause stoppages. The cuts averaged 10.42 ft. high by 130 ft. long. A study of several logs indicated that during only about 15 per cent of the cutting time was the saw cutting to its full efficiency; that is, during about 85 per cent of the time the saw was bedding in at the ends and not cutting well in the center, or the ends were below the floor line and only part of the wire was doing effective cutting. This does not mean that the wire was cutting to only 15 per cent of its capacity, but it means that during about 85 per cent of the time part of the cutting was being wasted. If higher ledges had been worked, the wire would have cut to its full efficiency a large part of the time, as it would have had a greater distance to saw after it had bedded into the ledge before the ends went down below the floor level. The exact saving to be expected cannot be calculated, but it would be large if really high cuts were made.

All the prime costs, including depreciation on equipment, were carefully kept, with the idea of determining what can be expected from wire saws. The actual figures obtained are presented in the accompanying table. Forecasts follow of possible improvements in each of the separate items of cost listed in the table.

On the basis of the data obtained in the tests, an assumption that a saving rate of 166 sq.ft. per hour can be maintained seems reasonable. At this rate the labor charge would be 1.108c. per square foot.

Average Costs of Wire-Saw Operations

	Distribution of Average Cost, Cents per Square Foot
Labor	5.89
Sand	2.07
Wire	3.09
Repairs	1.26
Miscellaneous	0.93
Power	0.35
Depreciation	0.35
Total	13.94

On a ledge laid out for proper drainage, the sand could be used until discarded by a classifier. Also, on a high ledge, the direct loss of 20 per cent due to cutting below floor level would be greatly reduced. It is impossible at present to figure accurately the saving possible if both these factors are considered. However, it seems probable that 50 per cent reduction in this item may be effected. At this rate, the sand charge would be 1.04c. per square foot.

The wire consumption constantly grew less, owing to the fact that the safety

factor in wire life was being cut down. On the last cut, 3,500 ft. of wire made 3,434 sq.ft. of effective cut. Wire costs $19 per 1,000 ft., or 1.9c. per foot. If the wire is good for a square foot of cut to the running foot of wire, the cost should be 1.90c. per square foot.

The repair bill was high (1.26c. per square foot) because we were just feeling our way and were working in a place not well suited to wire saws. Repairs should not cost more than 1c. per square foot.

Other charges, totaling 1.63c. per foot, will change little unless softer stone is cut.

Adding these revised costs, we find that in operating the saws properly on similar hard stone we should be able to cut the costs to about 6¾c. On ordinary stone, it should be possible to cut this in half again, thus finally coming to a cutting cost of about 3½c. a square foot.

After the wire-saw experiments were finished, the second floor of the same stone was worked with channelers to gain a cost comparison. This comparison is not entirely fair, because the channelers were fully developed machines, in the hands of an expert crew, operating on a ledge with both ends open for drainage, and ideal in every way for operation of a channeling machine. Also, the machine men felt that their jobs depended on making speed, as they knew that the work was being checked against the previous work of the wire saw. Under these conditions, the channeling prime cost was 15c. per square foot cut.

As mentioned before, the wire saw was at a disadvantage in being used on a shallow ledge and in being in the hands of an inexperienced crew. Also, there were only one-half enough wire saws to keep the men occupied. In spite of these handicaps, the wire saw cut for 11.22c. per square foot, as against 15c. per square foot by the channeling machines.

From the analysis of costs, it seems safe to say that wire saws, properly operated, will cut for about one-half the cost of channelers. As channeler cost usually is figured at about 8c. per cubic foot of stone produced, the savings should be about 4c. per foot. Applying this figure to the 1930 production, as reported in Department of Commerce Statement No. 9632, released June 9, 1931, we arrive at a possible saving during 1930 of about $550,000 in quarrying costs. In addition to this, successful application of wire saws should result in a substantial reduction in waste, which, when applied to the 1930 production and value figures, would have been worth some $720,000 in that year. Thus the total saving to be expected from successful application of wire saws in Indiana is well over one million dollars per year.

Minerals of the Sillimanite Group

Fifteen years ago, andalusite and other related minerals of the sillimanite group were interesting only to students of mineralogy. New developments in the refractories field vested them with commercial possibilities. A demand for economic deposits resulted, and from a large body of the mineral named, occurring in California, one manufacturer has produced since 1919 andalusite in quantity sufficient for 350,000,000 spark-plug cores—an episode that sheds a glamour around all museum specimens as yet unused by industry. This paper, by the director of research for the Champion Porcelain Company, summarizes admirably the economic position of the sillimanite group today. Besides andalusite the only other members of the group to have won as yet a niche of promise for themselves, by reason of their availability for the moment in commercial quantity, are cyanite and dumortierite. The former, though common in small tonnages in many localities, is being mined today principally in Arizona and in India, occurring interestingly in the latter instance in the State of Kharsawan as stray boulders of a source unknown. Cyanite is also being produced in North Carolina, as yet in minor quantities. So little of a positive nature is known about the size of the reserves of all these minerals that prediction is impossible as to what would happen should demand increase.

F. H. Riddle

Many of the products that are being made from ores of the sillimanite group, and synthetic substitutes for them, have unique properties, and service tests prove that they play a major part in improvements in metallurgical processes. About 1917, ceramists at the U. S. Bureau of Standards, given the problem of developing a better spark-plug core, decided to experiment with sillimanite; but, because of the scarcity of the natural minerals of this group, they decided to prepare synthetic sillimanite. The successful use of synthetic sillimanite caused manufacturers to search for the natural mineral in commercial quantities.

Among the minerals having the chemical composition $Al_2O_3.SiO_2$ are andalusite, cyanite, and several forms of sillimanite, including chiastolite, fibrolite, bamlite, zenolite, and worthite. These are spoken of as minerals of the sillimanite group. Dumortierite, although containing more alumina and some fluxes, is similar in many ways.

In 1919, Dr. Joseph A. Jeffery, president of Champion Sillimanite, Inc., located a deposit of several million tons of andalusite in the Inyo Range, Mono County, Calif. This deposit has been described by Peck and others. Because the apparent specific gravity of the ore varies directly with the silica and alumina content, it is mined and blended according to Jolly-balance tests run at the mine at frequent intervals. The minerals associated with the andalusite include lazulite, pyrophyllite, muscovite, corundum, rutile, pyrite, barite, and lazurite. None of these are objectionable in porcelain cores for spark plugs. For refractories the fluxes are more objectionable, and the mining is done accordingly.

At the Champion Porcelain Company's plant, in Detroit, the ore is treated by one of two methods, depending on the use to which it is to be put. For refractories, it is crushed to minus 1 in. in a jaw crusher, and ground in a Herman ball mill. The crushed ore is sized by screening and stored. When andalusite is used in porcelain and bodies of similar type, fine grinding is essential. The crushed ore is reduced to 8 mesh and finer in a Hardinge ball mill, passed over a magnetic separator, and then ground in a pebble mill to minus 30 mesh, 60 per cent being finer than 325 mesh.

The andalusite ore is of particular value because it has no appreciable volume change when converted to mullite. This makes possible its use without previous calcination, which has advantages.

Cyanite, although common in some Southern states, occurs in quantity only in one or two localities, where it has to be concentrated. A large quantity of ore has been mined in Ogilby, Ariz., by the Vitrefrax Corporation, of Los Angeles, in California. All of the cyanites mentioned by the author appear to have an extremely platy structure; and, when calcined, are likely to be friable and difficult to bond. Calcination is necessary before they can be used in any great percentages.

The Ogilby ore used by the Vitrefrax Corporation is of particular interest because of an ingenious method developed for treatment. It is heated in a rotary kiln so as to reduce the iron oxide for later removal by magnetic separation and convert alpha to beta quartz. The calcine falls from the kiln into water, for quenching and rapid chilling, which converts the quartz back again to the alpha form. This quartz conversion causes a sudden expansion and a like contraction, while the cyanite

shows only a slight expansion and contraction. The quartz is shattered loose. The ore is then crushed in a swing-hammer crusher and washed over slightly inclined shaking screens. The relatively pure cyanite fibers will remain on the screen and the quartz gangue will pass through as sand. This separation method is also used by Vitrefrax Corporation for refining pyrophyllite and some other aluminum silicates. Refined cyanite is used in several ways, depending on the product desired.

A more massive cyanite is being imported from India into the United States by Chas. F. Taylor & Sons Company, of Cincinnati, Ohio, which is known under the trade name "P. B. Sillimanite." The rock is chiefly cyanite, with some sillimanite and occasionally andalusite. The ore is shipped in the original boulder form to Cincinnati, Ohio, where it is calcined.

Sillimanite appears to be much scarcer than other minerals of the group. Several small outcrops have been opened in the United States, chiefly in South Dakota. It has been discovered in India, where it is said to occur in large amounts.

A deposit of dumortierite was opened in 1925 by Champion Sillimanite, Inc., and used exclusively by the Champion Porcelain Company. It is in the Humboldt Range, near Oreana, Nev. The ore contains impurities such as quartz, paragonite, and muscovite mica. Over 2,500 tons has been mined, shipped to Detroit, and treated in the same manner as the finely ground andalusite.

Synthetic mullites have been developed to a point where they are of equal importance with the natural minerals for use as refractory products. The Vitrefrax Corporation, of Los Angeles, under the direction of Thomas S. Curtis, has made many interesting developments. It has produced mullite by the complete melting, in an electric-arc furnace, of various alumina-silica mixtures, forming "pigs" which are crushed to grain and fabricated in the usual way. The company also has a Curtis converter process which is much more economical to operate and which produces a more granular mullite than the cast brittle product. Refractory products manufactured by this company are marketed under the trade names "Argon" and "Durox." It also markets other products, including the partly converted cyanite described in a preceding paragraph.

At present, cost is the limiting factor in the almost universal use of sillimanite wherever fireclay, silica, high-alumina, or kaolin brick are used. In the super-refractory field, costs are about the same. Recent developments indicate that processes will be perfected that will make possible the production of sillimanite refractories at much lower prices, and permit their much wider use.

Earth Resistivity Survey in Illinois

Problems in geophysics lending themselves most readily to solution by earth-resistivity methods were worked out by Mr. M. King Hubbert in Illinois last summer. They comprised the location of faults in Paleozoic sediments, of gravel deposits and ground-water supply in glacial drift, and of anticlines buried beneath the latter. Work proposed on a problem in connection with the lead-and-zinc-bearing area of northeastern Illinois was deferred because the preliminary study suggested doubts that many major deposits remain in the known productive area. For this work the author used the so-called Megger Ground Tester, the essential elements of which are a hand-cranked direct-current generator, a double commutator system, and an ohmmeter that reads directly in ohms the resistance of the block between the two potential surfaces passing through the points of attachment of the instrument's two potential electrodes. According to the author, the instrument is light, compact, durable, and simple and convenient to operate. He holds it capable of supplanting much investigational work. Its use was demonstrated at the annual meeting of the Institute a year or two ago. Mr. Hubbert is an instructor in geophysics at Columbia and associate geologist of the Illinois Geological Survey.

M. King Hubbert

During 1931, the writer of this paper was asked by the Illinois State Geological Survey to make a study of some of its economic geological problems, with regard to the applicability of geophysical methods of prospecting to their solution. The problems presented themselves in connection with (1) the fluorspar area of Southern Illinois; (2) oil structures in the south-central part of the State; and (3) the water supply and gravel deposits of the glacial drift. The prime requirements were to find a means of obtaining a maximum of useful information with a fairly modest outlay of expenditure for technical staff and equipment. A preliminary study was made of each problem by means of available office data and published reports with supplementary field reconnaissance where necessary.

Any geophysical prospecting for fluorspar in Southern Illinois would have to depend either upon the physical properties of the ore and associated minerals, or upon the properties of the geologic structure containing them. Detection of associated structures seemed to offer most promise. Inasmuch as the vein deposits occur as mineralization along fault planes, the location of unknown faults would afford a step toward the location of deposits of the mineral. For this purpose either earth-resistivity or seismic means might be applicable. Because of its greater simplicity and less expensiveness, trial by the former was recommended.

The petroleum of Illinois is obtained mostly from anticlinal structures in the southern half of the State. Earth resistivity and seismic methods seemed the most likely to be successful, so decision was made to try the former first.

As for the water-supply problem, water supply for domestic uses in Illinois has to be obtained largely from glacial drift. The problem in this instance resolved itself into that of finding gravel deposits hidden in the glacial drift. Earth resistivity seemed the most promising means of investigation.

Apparatus used was the Megger Ground Tester, manufactured by Evershed and Vignoles, London, England. This instrument lends itself to the determination of the specific resistivity of a semi-infinite homogeneously conducting medium by the method worked out by Wenner. Wenner showed that if on the surface of a homogeneously conducting medium having a plane surface and infinite extent on one side of that plane, four electrodes in the same straight line and having an equal spacing, a, are placed, then $\rho = 2\pi a \frac{E}{I}$ where ρ is the specific resistivity of the median, E the potential difference between the two inner electrodes, and I the current between the two outer electrodes. Gish and Rooney developed the technique of resistivity prospecting for inhomogeneous media by considering ρ of Wenner's formula to be merely the apparent specific resistivity, and letting the value of ρ serve as an index of the variation of resistivity as a function of the electrode interval a and the position and azimuth of the lines of the electrodes.

The point midway between the two potential electrodes was (1) the position of the station; (2) the electrode separation a; and (3) the azimuth α of the line of the electrodes. The apparent specific resistivity ρ is measured as an empirical function of any or all of these variables. If ρ is measured as a function of position of station while both a and α are kept constant, the resulting curve indicates the approximate position of the bodies having marked

differences in conductivity. If at any particular station an approximate horizontal homogeneity of resistivity is discernible, a curve of ρ measured as a function of electrode separation a gives variations due to a change of resistivity with depth.

Determination of ρ as a function of the azimuth α while keeping the position and the electrode separation a constant will indicate both the presence of major horizontal inhomogeneities and to a great extent the orientation of the body exhibiting it.

The essential problem of prospecting by this method is to determine by means of combinations of the above-mentioned function of three variables the horizontal position, depth, and, as nearly as possible, nature of bodies near the earth's surface differing markedly with respect to specific resistivity from their surroundings.

Three types of problems were investigated by the earth-resistivity methods specified—(1) faults in Paleozoic sediments; (2) gravel deposits and ground-water supply in glacial drift; (3) anticlines buried beneath drift.

The investigators found that faults could be discovered, especially if the outcrops on the opposite side were different or if a considerable shear zone were associated with the fault plane. Some minor faults shown on the map, with the same formation outcropping on both sides, were missed.

Major gravel deposits having fairly sharp boundaries, even if full of water and buried under considerable till cover, could be detected readily because of their relatively higher specific resistivity. Moreover, the one buried anticline tested gave a pronounced anomaly, which coincided with the anticlinal axis as independently determined from well-log data. Attempts at precise depth determinations have not as yet been successful.

For problems of this nature, therefore, this method of investigation seems, in spite of its limitations, to have fairly demonstrated itself to be a highly useful and economical means of at least doing reconnaissance work, and to be capable of supplanting a great deal of more expensive means of investigation.

Katanga Hydrometallurgical Operations

After demonstrating by laboratory tests the possibility of recovering copper from the lower-grade Katanga ores by leaching and electrolytic deposition, Mr. Wheeler recommended construction of an experimental unit with a monthly capacity of 100 tons of cathodes. Results obtained in this plant in 1921-22 under the direction of Mr. H. Y. Eagle were so satisfactory that a plant with an annual capacity of 30,000 metric tons of wirebars was authorized. Plans were submitted in 1923, but the work was not sanctioned until 1925. A combination of circumstances delayed construction, which otherwise could have been completed in two years, with the result that the first unit of the leaching plant was not started until April, 1929, when it was put in operation by a locally organized crew, with Mr. Eagle acting in an advisory capacity. When conditions in the copper industry forced the plant to curtail in December, 1929, an ability to meet promises made as to capacity, extraction, acid and power consumption, and overall operating costs had been satisfactorily demonstrated.

Archer E. Wheeler

H. Y. Eagle

Among the numerous features that distinguish the principal hydrometallurgical processes now used at Katanga from existing wet processes for recovery of copper may be cited. Continuous operation; agitation instead of percolation; continuous washing of sand tailing in counter-current multi-deck classifiers; continuous purification by neutralizing part of the solution with ore.

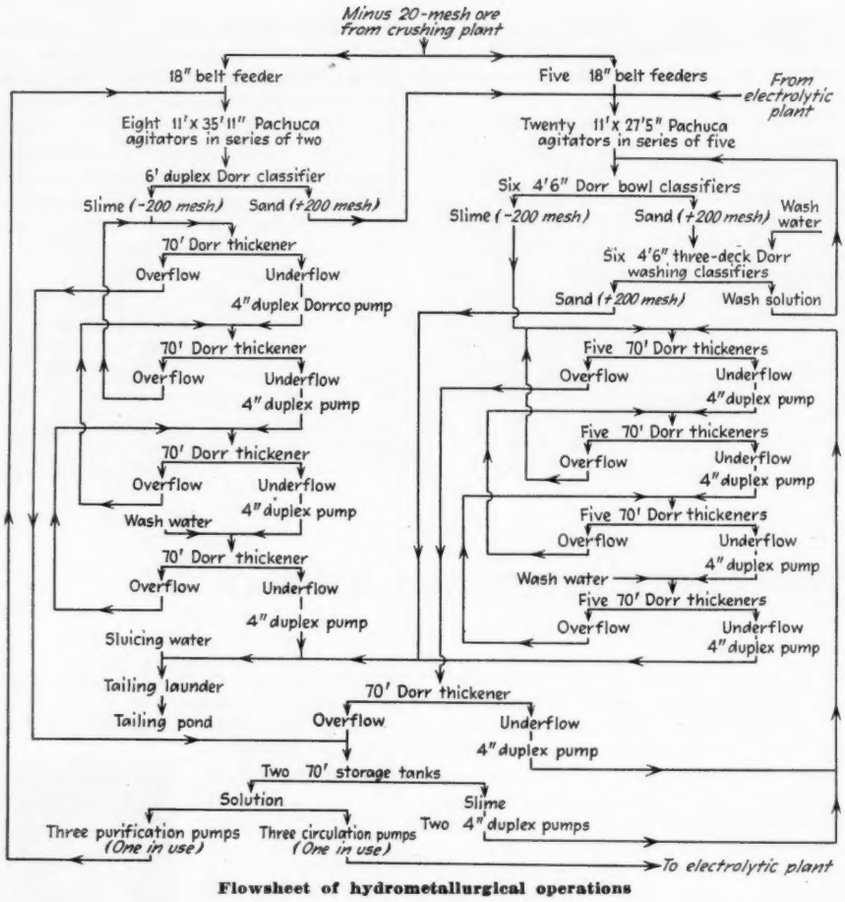
Numerous difficulties were encountered in the early stages of the work, primarily caused by inability of the agitators to handle the 35-mesh feed; by inability of the vacuum filters to handle the slime tailing; and by failure of so-called acid-proof equipment in the corrosive solutions.

A simple attachment on the agitators solved the first difficulty so effectively that grinding to 20 mesh proved sufficient. Filtration was discarded and counter-current decantation in Dorr thickeners adopted. Corrosion was overcome by the use of lead, duriron, rubber, asphalt, mastic, glass, and porcelain. Several nickel-chromium alloys were later added to the list.

Primary crushing equipment, housed in a steel-frame structure, 45x47 ft., with four operating floors, consists of a 48x36-in. jaw crusher and three 5x12-ft. trommel screens; the secondary crushing plant, also a steel-frame structure, 40x57 ft., arranged on five floors, houses two No. 5 gyratory crushers, two 5x12-ft. trommel screens, and two sets of 54x16-in. rolls. All this equipment is of Allis-Chalmers manufacture. Crushed ore is delivered by belt conveyors to the ore storage, a steel-frame structure 198x304 ft. with three bays and two operating levels. Ore reclaimed from the pipe with a Robins Standard Messiter reclaimer is delivered by conveyor to the dryer bins. Merrick weightometers are provided wherever tonnages handled by conveyors are recorded. Six wood-fired 7½x55-ft. double-shell Ruggles-Coles dryers, five 6x12-ft. Allis-Chalmers rod mills, and ten 8-ft. Hum-mer screens comprise the grinding and screening equipment. Ground ore is delivered to 2,000-ton steel bins, with bottoms of the catenary suspension type. All conveyors throughout the plant are equipped with Stephens-Adamson ball-bearing idlers and Goodyear or Goodrich belting.

The agitator building is a 102x115-ft. steel-frame structure with three operating levels. Standard Pachuca vats of steel, lined with 10-lb. lead, operating with 30-lb. air, are used. They are arranged in five acid sections of four in series and one purification section with two parallel rows of four in series. Return solution from the cell room is distributed to the acid sections by a five-compartment weir, joining the ore in a mixing box ahead of the first agitator in the series. Enough solution is pumped to purification from the first storage thickener to maintain a total iron content of 6 grams per liter.

The classifier building, a steel-frame structure 109x99 ft., contains six sections, each consisting of a primary bowl classifier followed by and integral with a three-deck washing classifier. Sand is progressed successively through the four decks. Water is added in the last washing deck, and the overflow from the bowl classifier added to the feed wells of the primary thickeners. Classi-



Flowsheet of hydrometallurgical operations

fier tanks are of concrete, with 2-in. mastic lining, mechanisms of lead-covered steel, and rake blades of duriron.

Arranged in five horizontal benches, in 3 ft. 4 in. steps, are 28 70-ft. Dorr thickeners. The first four rows, in descending order, serve as third wash, second wash, first wash, and as primary thickeners, respectively. Of the four thickeners in the bottom row one is a spare, to hold the contents of any other thickener during repairs; one is used to clarify the overflow from all primary acid thickeners, and the last two serve as storage tanks. The first vertical row serves as purification thickeners; the next five are acid thickeners for the corresponding agitator and classifier sections. Each thickener is roofed individually, and all are connected by walkways over the top. They have concrete bottoms, steel sides, and 8-lb. lead lining; mechanisms are of lead-covered steel with wooden extension arms. Access to the cast-lead cones, valves, and piping is provided through tunnels under each vertical row, terminating in a sump and pump house. Dorrco pumps lift the spigot discharge uphill from one tank to the next, the solution overflowing in the opposite direction. Wash water is added in the last thickener to compensate for solution entrained in the tailing. The feed to the five acid thickeners is distributed by a five-compartment weir and launder system. All launders are of wood, lined with 6-8 lb. lead, with wood lining to protect the lead when necessary. Tailings are laundered to a gully below the plant site, which leads to a pond.

Two pump houses have been provided, one to pump the solution from the last storage tank to the cell room, the other to pump solution from the first storage tank to the purification agitators. Duriron centrifugal pumps are used, 8x6 in. for the former service and 6x4 in. for the latter and for the sumps in tunnels and agitator basement. All piping is lead or lead-lined steel for high heads.

The electrolytic tank house is a steel-frame structure, 266 ft. 6 in. by 405 ft. 0 in. in plan, sheathed with asbestos-protected metal and roofed with a wood deck covered with built-up roofing. This building is arranged with four crane aisles with a crane transfer at one end, to provide means for shifting cranes from one aisle to the other. The main operating floor is all on one level, and a basement is provided with ample headroom for tank inspection and launder installation. There are 160 electrolytic tanks in the building, each 62 ft. 6 in. long, 3 ft. $2\frac{1}{2}$ in. wide and 4 ft. $2\frac{1}{2}$ in. deep inside. The tanks are of reinforced concrete lined with 2 in. of asphalt mastic, and are equipped with a feed pipe at one end, an overflow dam at the other, and a hard lead plug and seat in the bottom for draining and cleaning out. Solution pumped from the leaching division is split into two equal parts by a weir tank in which the sulphuric acid requirement of the process is added continuously. Solution from the weir compartments is laundered to two constant-head tanks, from which hard-rubber header and riser pipes distribute the feed to the individual tanks. Overflow solution from the tanks is collected by a launder system in the basement, and laundered through a tunnel to the agitator section of the leaching division. Conductor bars 2x6$\frac{5}{8}$ in. in cross-section are supported on wood insulating strips on the tank walls. Electrodes are arranged in three separate electrical groups in each tank, the groups being in series and the electrodes in multiple. The entire tank house is laid out in two main electrical circuits, each taking normally 8,000 amp. at about 460 volts. The center point of each electrical system is grounded through a low resistance in order to protect the workmen from injury. Sixteen of the electrolytic tanks are equipped so that they can be fed with a separate circulation of pure solution, and enough of these tanks are so used to produce the necessary starting sheets from soluble anodes. The remainder of the tanks are equipped with insoluble anodes of hard lead. Electrodes are handled to and from the tanks by 5-ton tank-house cranes of the stiff-leg type. Forty-eight-hour starting sheets are stripped on racks at the end of the building, looped on standard Morrow clip machines, flapped on tables and suspended in racks ready for charging by the crane. Finished cathodes, 10 to 14 days old, are drawn from the tanks by the crane, spray-washed, and unloaded on cathode loading machines of the Great Falls type, which deposit them in piles on cars going to the furnace refinery.

The furnace refinery building is a steel-frame structure, 124 ft. 8 in. by 234 ft. 0 in. in plan, sheathed and roofed with corrugated galvanized steel, and arranged with two operating levels. It is equipped with two 130-ton refining furnaces, each provided with a 38-ft. diameter casting wheel of the Clark type, together with bosh and inspection conveyors. The charging bay is served by a 7,000-lb. charging crane, made by the Morgan Engineering Company, and the casting bay by a standard 5-ton crane. An outside runway equipped with a standard 5-ton crane provides handling facilities over the storage and shipping yard. The furnaces were originally designed for grate firing, but on installation were equipped for pulverized-coal firing with Heyl and Patterson burners and feeders. The furnaces are built with air-cooled brick bottoms.

Smelting Raw Concentrates at Miami

Changing trends in concentration resulting in a high-grade flotation product present a real problem to the copper smelter. Handling difficulties are numerous and losses may be great. Where preliminary roasting is required, the only alternative is to follow orthodox practice, taking all precautions to avoid dust losses. Where roasting is unnecessary or undesirable, raw charging becomes a possibility. In the original Miami plant of International Smelting Company all material was dried in Wedge furnaces prior to the reverberatory smelting. At times a partial roast was required. With over 80 per cent flotation concentrate in the charge, this was a dusty operation, requiring a Cottrell treater and specially designed hoppers and calcine cars. In the furnace itself dusting was a severe problem, arch erosion was rapid, and operation of the waste-heat boilers difficult. Wet charging, successfully developed by A. D. Wilkinson at Cananea, was tried in 1927, with promising results, and the practice has been continued and developed. The cost compares favorably with that made in smelting a dried charge, and where conditions warrant adoption of the method many benefits may be gained. Elimination of the dusting problem greatly simplifies operations and improves working conditions.

P. D. I. Honeyman

At present the reverberatory-furnace charge at Miami is made up of 77 per cent concentrates, 4 cement copper, 7.5 flux, and 11.5 per cent secondaries, with an average analysis of 33.27 per cent copper, 24.82 sulphur, and about 11 per cent moisture, though charges with about 15 per cent moisture have been treated. With increasing moisture content the charge becomes sticky and difficult to handle. The usual calcine cars could not be used for this material, but 60-ton concentrate cars, of the bin type, with removable bottom of 2x8-in. board, were available in sufficient number to permit unloading all concentrate on day shift, leaving enough stored charge in cars for the ensuing 16 hr. Further simplification has been achieved by bedding directly in the cars.

The plant has four reverberatory furnaces, 120 ft. long, one 25 ft. wide and three 21 ft. Only one is in operation at present. Designed for calcine, the furnaces are equipped with drag chain conveyors running from bins under the feed floor and discharging to the furnace through gate-controlled feed pipes. This system, with certain modifications, worked fairly well for wet feed and has been retained after an attempt to use a belt conveyor and tripper car resulted in a higher fuel consumption. Furnaces are now equipped with charge bins under the feed floor, feeding to the drag conveyor with a pan feeder; 8-in. charge pipes, spaced at 44-in. intervals are controlled by gates from the main runway. Peepholes in all charge holes permit observation of the charge, and the flow may be assisted by working a rod through the peephole. Heavy scrapers serve to keep the chain clean by reducing the amount of "drag-over."

In charging, the car is spotted over the furnace bin, all slides are opened, and feeder and conveyor started. Balanced draft prevents appreciable infiltration of cold air through the charge holes. Starting from the firing end, the charge is fed into two or three holes until they are full, continuing in this manner until the entire charging zone, about 65 ft. down the furnace, is filled, an operation requiring 20 to 30 min. Best results were obtained with large charges about six times a shift. A deep molten bath is maintained between the two charge piles in the smelting zone, and every effort is made to get as much charge as possible down to the surface of the bath. Failure to do so slows up the furnace. Little trouble has resulted from explosions as the charge enters the bath. Some shooting has taken place, particularly if the slag layer is not sufficiently deep. A rate of 600 wet tons per day has frequently been obtained—more than 30 tons per charge. Liquid converter slag is fed through a launder at the middle of the bridge wall. At times this causes violent boiling of the bath extending through the entire length of the charging zone. This assists materially in mixing the surface charge and is also the one visible sign accounting to some extent for lower accumulation of magnetite.

Gross oil consumption per ton of dry solid is higher than for a dry charge, but this is offset by increased waste-heat recovery and by saving of the oil used in dryer operation. For six months in 1931, treating wet charge, the gross fuel ratio was 0.902 bbl. per ton dry charge, compared with 0.667 in 1924, with dry charge, and 0.577 in 1926, with partly roasted charge. Equivalent waste-heat recoveries during the same periods were 0.470, 0.263, and 0.219 bbl., respectively. Fuel consumption for drying in 1924 was 0.105 bbl.; in 1926, 0.077, giving a total net fuel ratio of 0.432, for 1931; 0.499 for 1924; and 0.435 for 1926.

A battery of Cananea low-pressure oil burners is used, with 40-oz. primary air. Secondary air is closely controlled by adjustment of auxiliary air ports in the bridge wall. The furnace is run on low draft, about 0.02 in. of water, to reduce to a minimum infiltration of air. Much difficulty was experienced in establishing proper conditions until the size of the furnace uptake was materially increased, to 16 ft. width and 82 sq.ft. cross-section. Adjustment of boiler dampers insures a steady draft during charging and smelting.

Waste-heat recovery is high, 52.1 per cent of the fuel burned, against 39.1 for dried charge and 37.9 for partly roasted charge, and the small amount of dust simplifies the problem of keeping the boilers clean. Sulphur elimination varies from 25 to 30 per cent. The second figure corresponds closely to that obtained with a dry charge; 85 per cent of the free-atom sulphur is driven off.

To avoid the danger of tapping through the side wall under a pile of wet charge in the smelting zone, and to aid in keeping the front clear of magnetite, tap holes were first installed well toward the front, right back of the verb arch. At present matte is tapped through the front wall into ladles on a transfer car spotted on the inner track of the slag tunnel. This has been found highly satisfactory, and the method has been retained, although tapping through the charge pile, in the smelting zone, has later been found practicable. The matte flows more freely at the front, and circulation of matte toward the front plays a large part in keeping the bottom free from accumulations. It also allows skimming through the side back of the verb arch.

The furnaces are of the regular type, constructed largely of silica brick. A six-course band of magnesite brick is, however, built into the walls from the end of the charge line to the front of the furnace, and the front wall from the foundation to above the normal slag line is also faced with magnesite, as is the bridge wall along the slag line.

Utilizing Gravity
In Mining and Sorting

Description of Underground Practice and Technical Control at Butte, Montana, Developed at the Properties of the Anaconda Copper Mining Company

C. M. Harrer

WITHIN the past twenty years, the mining methods applied at Butte have undergone a gradual change. Considerable thought has been given to the maximum application of the principle of gravity as an aid in facilitating the removal of ore from the stopes, and the introduction of filling. Formerly, and chiefly on narrow veins, the practice was to carry long, continuous stopes, and stopes in series, on a more or less horizontal plane. This practice was discussed in my article "Square-Setting in the Butte District," that appeared in the July 30, 1927, issue of *Engineering and Mining Journal.*

In some stopes, early methods necessitated the use of the "waste car" and a corps of men engaged in laying track, general repairs, and in distributing waste filling. They also required the employment of extra equipment such as cars, rails, spikes, and timber, which could be used to greater advantage elsewhere. In some of the older stopes, cars and wheelbarrows were used to transfer ore from its resting place in the stope to distantly placed chutes. At present, many of the stopes are on the plan of the inclined-cut-and-fill system, locally classed under three types as timbered rills, rills stopes, and inclined square-set stopes.

In a square-set stope, illustrated in Fig. 1, the working breast is carried up by offsets. Broken ore falls onto slides on the filling and gravitates to within two sets of the ore chutes, an easy shovel's throw, and also allowing it to be readily sorted of its coarse waste content. This insures a cleaner ore and a minimum of shoveling effort.

Application of the system to the rill stope is shown in Fig. 2. Two sets next to the chute are not filled with waste, but are fenced off from the main fill and act as a reserve receptacle for the waste sorted from the broken ore. This method of carrying inclined floors or slides facilitates shoveling, and, on completion of an inclined slice, filling is introduced into the stope by merely knocking out or blasting out the sides of the end raise chutes facing the stope. The waste runs into the stope from the raise chute to its angle of repose, the floors or slides being first removed.

The system of alternate stoping and filling has made advisable division of the stope blocks—both square-set and rill—between two raises into a double stope. Thus, one wing or side is undergoing filling operations while the other is in the production stage. Services of general repair men, to carry up the chutes and fence off the filling to be introduced, are required only intermittently, and the general repair crew is thereby considerably reduced. In many instances repair men in the stopes are unnecessary, the miners carrying on the total stope work. This procedure has, in fact, become quite general.

Formerly, this method of mining was thought to be applicable only to steeply dipping orebodies, and a hard, firm hanging wall was considered absolutely essential. At present, stopes are carried up with comparatively weak hanging walls by keeping up the filling and employing stulls judiciously. Ore passes and their manways are bricked up from footwall to hanging wall, with round, unpeeled blocks, and carefully chinked and tightened when heavy ground is expected. With weak walls, the closer filling may necessarily entail mining smaller slices, and therefore a closer

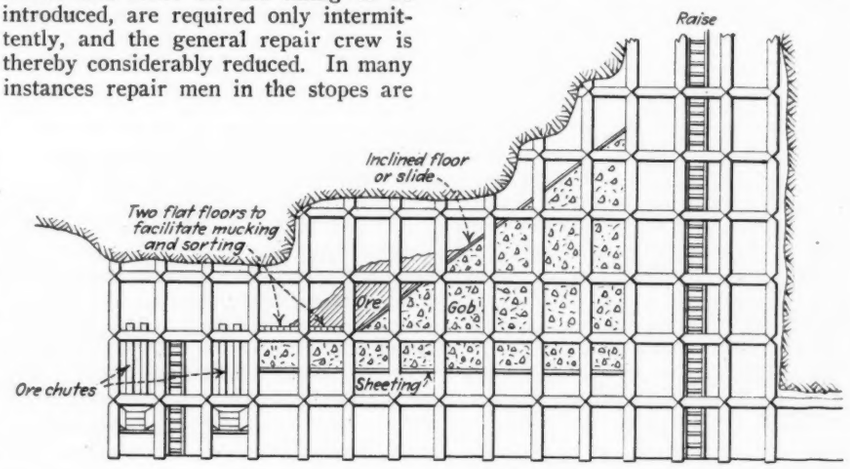


Fig. 1—A square-set stope of the inclined-cut-and-fill type

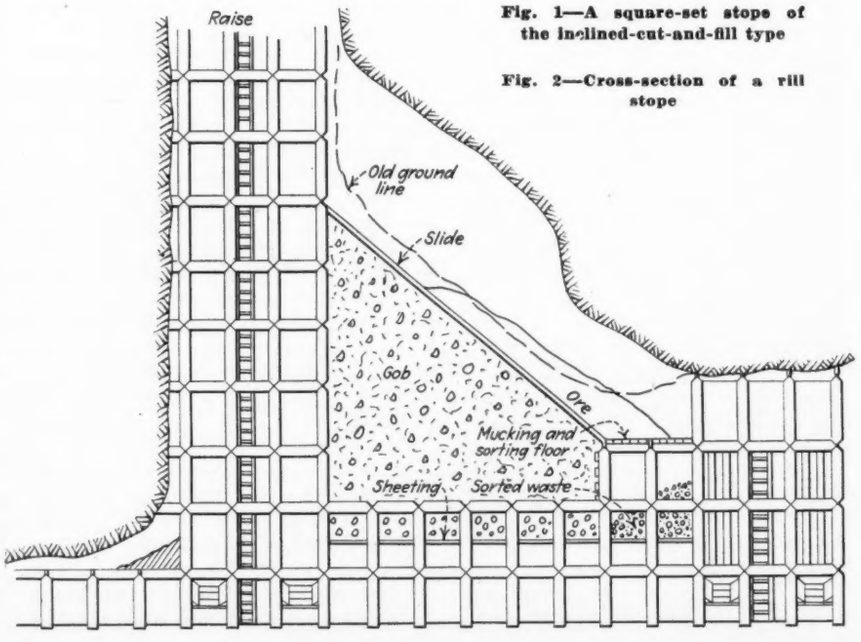


Fig. 2—Cross-section of a rill stope

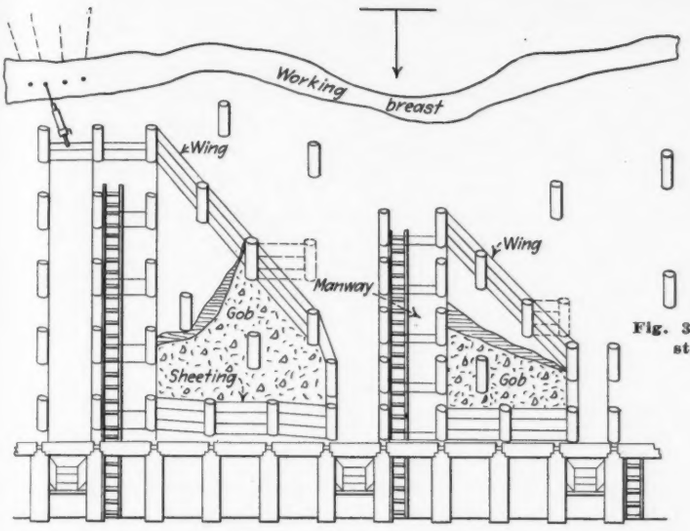


Fig. 3 — Sketch of stall stope

sequence of the operations of mining and filling. Where the hanging wall is quite weak, a system of connected timbering, square-set or stulls, is successfully employed as an auxiliary support. If the vein is comparatively wide, staggered square-set timbering is adopted. The inclined-cut-and-fill method has therefore a wide application at Butte.

In comparatively flat and narrow veins, the stull stope, illustrated in Fig. 3, is employed. Stulls with or without headboards are used for temporary support of the back. These stopes are all waste-filled. Wings are used to direct ore into the chutes and to separate it from the gob. All waste material sorted is thrown behind these advancing wings. This arrangement provides a ready means of waste disposal. All waste required for filling, besides that sorted directly in the stopes, is derived from level development work and sorting at the stations.

Initial sorting is in the stopes, where muckers remove a certain amount of the waste constituent while transferring the broken bulk of the vein matrix from the stope floor to the ore passes or chutes. Such material as is discarded is mainly the coarser fragments of obvious waste. The efficiency of this work is chiefly dependent on the number of sorting steps involved, supervision, facilities for ready waste storage, and character of the waste. Room for waste is always available in 90 per cent of the stopes. Waste that breaks finely is camouflaged by the darker ore minerals, and is neither readily discernible nor separated. Fragments of coarse waste, cut by stringers of ore, are either a source of loss when discarded or a source of dilution when included as ore. Cobbing is not feasible at Butte. *Thirty-three per cent of total excavation is sorted out as waste in all mine stopes.*

The second sorting of the ore is performed on the haulage level. This work has a number of variations, depending upon whether the levels are equipped with skip pockets or not. When the levels are so equipped, the closest sorting is carried on over the skip pocket. In this instance, two men are generally detailed to load a train of cars left by the motor crew. The stoping area is divided into sectors on the levels, and each sector is tapped by one of the two-man parties. One man manipulates the chute gates while the other "spots" the cars under the chute mouth and, as they are loaded, sorts any visible waste. This operation is essentially superficial, in that the waste sorted comes from the tops of the cars only, and the haste necessary in loading the trip does not permit anything but a cursory inspection of the waste content. In loading from slides or through "stop-boards" (Chinaman chutes), before permanent chutes are installed, this sorting is carried on with greater finesse, as the flow of ore is slower and more care can be taken.

When levels are not equipped with skip pockets, which condition is rare, various devices are employed to insure careful sorting at the loading points and in the stopes, little or no sorting being attempted at the level stations. In some mines, grizzlies in the stopes are constructed with 3-in. openings. Such grizzlies consist of 5x10-in.x6-ft. longitudinal timbers and 3-in. spacing blocks. Careful sorting is required of the muckers in the stopes. At times, where tonnages are relatively small, level sorting is carried on. A small portable grizzly, constructed of small rails or 2x4-in. boards, spaced 2 in. apart in the form of a grid, has been employed. This grid is supported over the cars, under the chute lip, by a staging. It can be removed readily as the occasion demands.

Sorting of development material is sometimes advisable, particularly in raising. Fig. 4 illustrates a method used in applying sorting in a raise. Obviously, sorting cannot be conducted on the mining horizon. To accomplish sorting without interfering with advance, a sorting pocket is installed. The miners may sort their own breakage, or an extra man may do it, depending on conditions involved.

Sorting in level headings is accomplished by the mucker, who permanently stows the waste along the sides of the heading or collects the waste in a pile and later loads it into cars marked for waste. When a broad band of waste occurs in a heading, resuing is employed. Where mechanical mucking is used in headings, sorting is not carried on at the breast but at the station.

On a level with skip pockets, the ore undergoes a third and final sorting at the station, before being hoisted to the surface. A gravity-dumping, sorting grizzly with bars 2 in. apart is placed over an orifice in the floor of the station. This grizzly consists of series of 1x2-in. steel bars held by spacers and tierods in the form of a plane grid. The sides and one end are boxed in by steel plates. This arrangement is so designed that it will hold one standard car of ore. Cars are dumped onto this grizzly one at a time, and one or two men carefully sort out the waste.

When the barren material is removed, the operator steps from the grizzly and releases a trip lever. The eccentric load on the grizzly—that is the ore on one end—rotates it through a limited vertical angle, and the ore slides into the skip pocket below. The grizzly is so balanced that, when empty, it returns to the horizontal either automatically or with the assistance of a slight pull. It is then locked in place by the operator, and is ready to receive another car of ore. Operators become very observant and proficient after a little instruction.

This method has two principal disadvantages where high-speed sorting is desired: First, the car of ore, when dumped, tends to pile up at the front end of the grizzly. The sorter must take a pick and distribute it to uncover fragments of waste. Second, the operation of sorting is intermittent. Only one car of ore can be dumped onto the grizzly at a time, and the sorter must take time to step from the grizzly and trip the

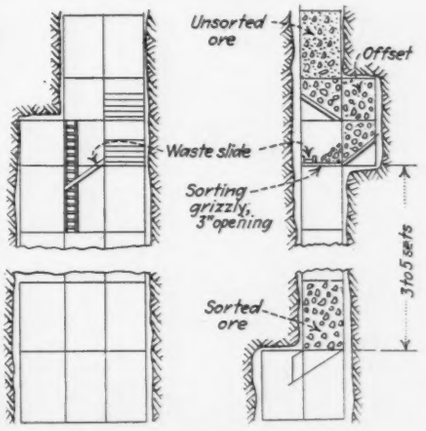


Fig. 4—Arrangement for sorting ore in raising operations

sorted load into the skip pocket. Large rocks become wedged between the dipping end of the grizzly and the side of the skip pocket; time is lost in freeing the grizzly. Some danger prevails that under high-tonnage operations sorting may be neglected.

The largest operator in the Butte district is the Anaconda Copper Min-

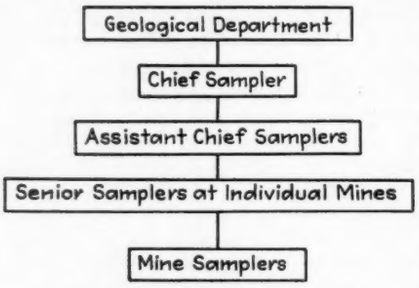


Fig. 5—Organization chart of mine-sampling personnel

ing Company. This organization, because of its many ramifications and the magnitude of its technical staff, exerts a strong influence on the technical growth of the entire district. Its methods of mining and technical control are considered standard practice in this district, and are generally adopted, with minor variations, dependent on the relative size of the companies.

The many phases of underground operations necessitate a close checking not only of ore in place but also of "ore widths," "mining widths" or "actual excavation," "the effect and control of dilution," "the smelter grade," and that source of anxiety: the gobs. A system of grade and mining control that permits checking and rechecking has been developed. An organization chart of this control is shown in Fig. 5.

In the control of sampling, the master link is the mine sampler. Each mine is divided into a series of "beats," each of which is covered by a trained sampler whose duty it is to take a series of representative samples of the various working breasts. These samples are assayed and recorded, with their widths, in such a form as to be readily interpreted. All samples are taken breast high. A stope card is used to record the sampling results. On it is also shown a sketch of the stope floors in their advance, positions of the various samples cut, and the active working breasts at the time of sampling. On a ruled part of the card are recorded identification numbers of the samples, dates of sampling, and respective widths and values of the samples. These cards are kept on file for every stope and stope-floor horizon, and form the basis for intelligent and efficient guidance of mining operations, and for estimations of probable grade and tonnage. When conditions of walls are designated on the stope cards, one can readily determine whether waste broken is the result of careless mining, or attributable to natural causes, such as sloughing of gouges and caving of sericitized and hydrothermally altered sections of the hanging wall or footwall.

An important aid in efficient mining control is the "average card." On this card are recorded assay and mining widths of working faces. These cards, kept clipped to their respective stope cards, show assay and mining-width averages of every sample face. The assay width of a face is the width of all vein matrix determined as ore. The mining width of a face is the actual width of the final excavation at the sample face, or the assay width plus any barren width mined. It is generally determined from contract measurements. A check of these two widths, and their final percentage grades or "value averages," give an excellent picture of the efficiency of actual mining and also of the deleterious effects of dilution. The average card, in conjunction with the tonnage or car tally from the respective stopes, provides a means of computing the efficiency of extraction and the efficiency of sorting and also affords a check on gob material.

All stopes at Butte require filling, owing to the proximity of the numerous veins to one another, the character of the walls and the ore, and the dangers of unchecked subsidence. The major portion of the requisite filling is often obtained by sorting in the stopes themselves, or from stope development, and station sorting on upper levels, the waste being transferred down.

Practically all underground work in Butte is on a contract basis. A close check on the character of the material forming the gobs is therefore of vital importance. This check is carried on by systematic grab sampling and a visual examination of the sorted gob material performed by the regular samplers, with occasional checks by other samples. Results are recorded and acted on when occasion demands. All daily sampling records are made available to the operating officials every 24 hours, and form basis of judgment for the men directly responsible for extraction.

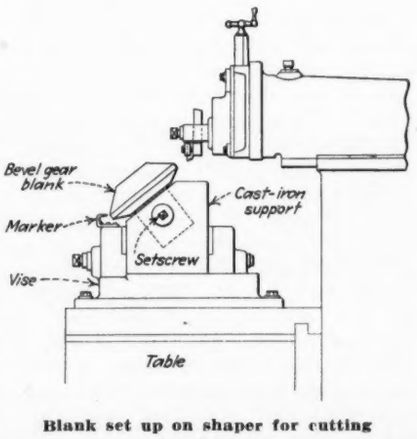
The degree to which methods for raising the grade of broken ore are applied is dependent chiefly on the time required for such work and the net gain therefrom. Operators installing a sorting system should not expect the same tonnage of sorted ore as they were accustomed to obtain of the lower-grade unsorted ore. Moreover, a cursory attempt at such operations, without any increase in the number of employees or changes in their distribution, leads to high-pressure work in the presence of officials, and little or no sorting in their absence, besides causing dishonesty and misunderstanding between employees and employers. The importance that such methods are to assume in mining operations should be determined on the basis of the economies achieved by handling a reduced tonnage and an improved grade of broken ore. Additional advantages include savings from handling less material in shafts, savings in rail transportation and treatment costs, and the fact that in filling stopes less material must be handled between levels and less labor is therefore required for distribution of filling.

I wish to acknowledge the courtesy extended me by Mr. C. L. Berrien, assistant general manager of mines, Anaconda Copper Mining Company, and Mr. George Vivian, chief sampler, Anaconda Copper Mining Company, both of Butte, Mont., in reviewing the data on which this article is based.

■ ■ ■

Machining Bevel Gears in a Shaper

CUTTING small bevel gears often presents a difficult problem in shops where suitable equipment for such work is lacking. J. A. McHugh, shop foreman at Magma Copper, Superior, Ariz., solved the problem successfully with the aid of the small shaper available at the shop. The setup shown in the accompanying sketch consists of a cast-iron fixture with an opening drilled at an angle coincident to the center angle of the gear, to accommodate the shank of the gear blank; and a small marker, bolted to the vise jaw, to facilitate setting and checking of the marked gear teeth. Pressure to hold the gear in place during the action of the cutting tool is applied by a countersunk setscrew threaded into the side of the jig.



Blank set up on shaper for cutting bevel gear

Improvements in Koepe Hoisting

A. Schurig

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Germany

HOISTING by sheave drive has gained increasing importance since its invention, in 1877, by Koepe. In England and the United States little attention has been given to this method of hoisting; in Germany, however, it has progressed with constantly increasing acceleration, and since 1910 the number of drum hoists in use has been steadily declining. In the Dortmund district, for instance, the number of Koepe machines in use has increased from about 180 in 1910 to 440 in 1928, whereas the number of drum hoists has decreased from 260 to 170 during the same period. Introduction of electric drive has been a material aid in this respect, on account of the smaller mass effect and the lower starting momentum, compared with the drum drive. The development has also been greatly influenced by the many improvements made in the sheave-drive system, such as the appreciable increase in the angle of contact in the Koepe-Heckel construction, and the better types of speed regulators, brakes, depth indicators, and sheave linings. All of these improvements aim principally at increased safety from rope slippage during regular hoisting operations. The fact that the rope, under certain operating conditions, is able to slip on the sheave, is unquestionably the weakest point in this type of equipment. One of the most important safety devices, the depth indicator, ceases to function under such conditions.

Although the danger from rope slippage is no longer as great as it was formerly—when it caused many operators, especially in the United States, to refrain from using this equipment—it is the reason why the deepest shafts in the world, where the Koepe hoist could be used to special advantage, are equipped with drum hoists. As the economical and mechanical advantages of the sheave-hoisting system increase with the depth, the value of this machinery in mines going to greater and greater depths is easily seen. Operators should therefore be especially interested in learning of a new type of sheave, the Karlik sheave, made by Heckel, Saarbrücken, which, to a great extent, has eliminated rope slippage. This clamp sheave, used extensively for years in all types of ropeway hauling, has now, as a result of this application, been adapted to the particular conditions encountered in shaft hoisting. It has a lifting capacity greatly surpassing that of any other known sheave. The capacity is, on the other hand, limited in such a manner that the rope slips in the groove whenever a definite, predetermined load is exceeded. This gives the sheave an advantage over drum hoists, because an overload caused by obstructions in the shaft or by "overwinding" will not cause the rope to break.

Most important among the characteristics which have given this sheave a reputation for safety is the maintenance of a constant clamp pressure as the rope stretches and becomes thinner. This also prevents any damaging action on the rope—the most frequent shortcoming of all older types of clamp sheaves. This pressure, a function of the pull on the rope, may be exactly determined. Another valuable property is that the clamp forms a semicircular groove which fits tightly around the rope and distributes the pressure over a large surface—the inclosed part of the rope exceeding 180 deg. Most types of construction provide only point

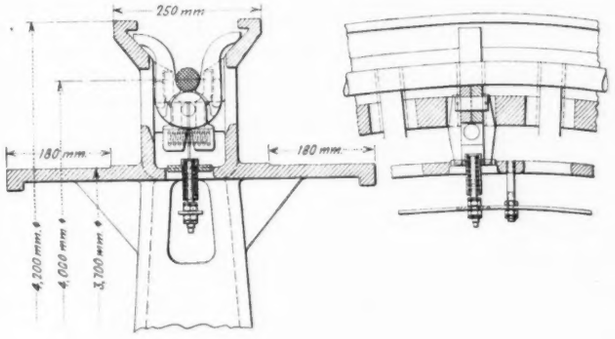


Fig. 1—Detail of 4,000-mm. Karlik sheave, showing clamp and rope

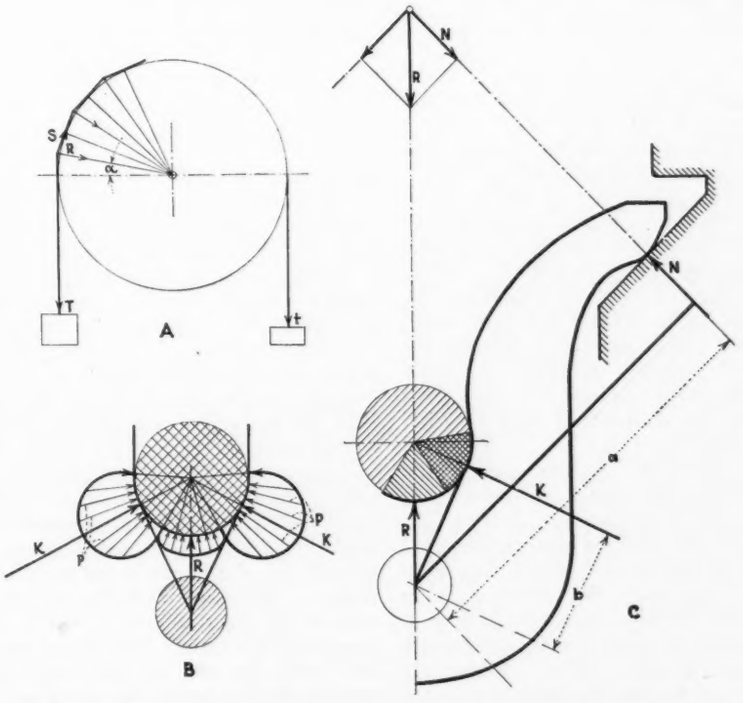


Fig. 2—Diagrammatic presentation of rope and clamp pressures

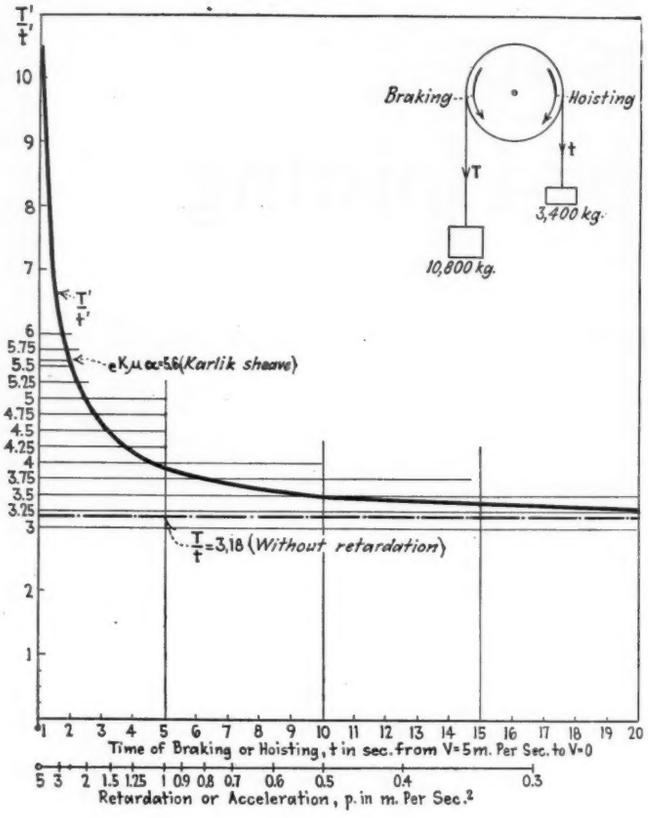


Fig. 3—Diagram showing T:t curve.

T = 10,800 kg.
t = 3,400 kg.
v = 5 m./sec.
Load retarded during braking, or accelerated during hoisting. The $\frac{T'}{t'}$-curve shows the change of $\frac{T}{t}$ caused by accelerating or retarding forces.

contact between rope and groove, with resulting high specific pressure, causing damage to the strands of the rope. Easy exchange of clamps, with practically no interruption in operations, and, most important of all, without taking off the rope, is another factor which greatly affects maintenance. With an ordinary sheave, equipped with a replaceable groove lining, the clamps, which are replaced every four to six months, can be changed only during a longer shutdown.

The statements made herein may be illustrated by an example from practice. Load on the two ropes of the sheave is: (1) T = 2,500 kg. (skip) plus 900 kg. (2 cars) plus 1,300 kg. (2 carloads) plus 5,600 kg. (1,000 m. rope, 40 mm. diameter, at 5.6 kg. per meter) plus 500 kg. (rope couplings), a total of 10,800 kg. (2) t = 2,500 kg. (skip) plus 900 kg. (2 cars) = 3,400 kg. The depth is 1,000 m. and the rope speed 5 m./sec.; the ratio $T:t$, giving the required hoisting capacity, is 3.18. Driving power exerted on the periphery of the sheave is $T - t$ = 7,400 kg., or 495 hp.

The diameter of the rope path, generously dimensioned to reduce the bending stress on the rope, is 4 m. As the rope is held in the clamps, these are kept from falling out of the sheave by guides supporting the lever ends in their outer position, as shown in Fig. 1. Completely to eliminate the slight noise from the clamps hitting against the guides, and to give the hoist operator a feeling of absolute protection as regards the clamps falling out, they are equipped with buffers and bolted to the sheave body. This special arrangement, used in shaft hoisting only, makes the exchange of clamps a little more difficult, but is an important safety measure wherever men are being hoisted.

The bolts of the clamp buffers are adjusted in such a manner that, even when the sheave is not in motion, the clamps open sufficiently to allow the rope to feed freely into them as soon as the sheave starts to turn. A relative adjustment of the clamp spring and the buffer spring causes the former to open the clamp sufficiently, while the latter spring keeps the clamp from reaching the guide, when the centrifugal force reaches a high point with the maximum number of revolutions. A low specific load on the springs may be maintained, to eliminate the possibility of a break. Should a spring break, in spite of this precaution, the safety of the operation is not endangered. If a clamp spring breaks, the clamp can open only wide enough to allow the rope to run through. If, in the worst case, the opening should be too small to permit the other rope to feed into the clamp, the rope will no longer touch the bottom of the latter. This reduces the clamp pressure, but the reduction is of no consequence, on account of the large capacity of the sheave. Break of a buffer spring is even less dangerous, as in such a circumstance the clamp, in opening, moves out until the end of the lever arm touches the guides on the periphery of the sheave.

Attention has already been drawn to the low rope wear caused by clamp pressure. This pressure is estimated as follows: Maximum clamp load is found at the point where the rope carrying the highest load T leaves the sheave. The forces are shown diagrammatically in Fig. 2, *A* and *C*. Tension T and the adjacent, somewhat smaller tension S produce a radial force $R = 1.95 \times T \times \sin\alpha$, T equaling 10,800 kg. and α = 1 deg. 52½ min. (96 clamps on the periphery), making R = 688 kg. From this figure must be subtracted the opposing force resulting from the centrifugal action exerted on the clamp (weight 4.5 kg.) or $C = \frac{1}{2} \times 0.1 \times 4.5 \times 5^2 = 5.6$ kg., making $R = 688 - 6 = 682$ kg. This gives a clamp pressure $K = 682 \div 2 \cos\beta \times a \div b$ = 1,530 kg., distributed evenly on both sides of the pressure point over the width of the clamp (3.6 cm.) as shown in Fig. *2B;* with a pressure surface of $2.1 \times 3.6 = 7.56$ sq.cm. the specific surface pressure on the rope is p = 202 kg./sq.cm. This surface pressure reaches a maximum under the pressure point and decreases with the distance from this point. The deviation from the average value is, however, too slight to affect the calculation.

A correct estimate of the calculated value of 202 kg./sq.cm. is possible only when a comparison is made with an ordinary hardwood groove. Calculated after Hertz's formula, a groove of 21 mm. radius, with a rope diameter of 40 mm., gives a maximum surface pressure of 373 kg./sq.cm., or 1.8 times the value found for a Karlik sheave. The surface pressure, 373 kg./sq.cm., is, furthermore, not a definite value, but rests on the assumption of a groove radius of 21 mm. If this radius is greater than half the rope diameter, as is usual, the maximum pressure will exceed 373 kg.; if smaller than 21 mm., the pressure will be correspondingly lower. Although the pressure cannot be determined with mathematical accuracy in the two instances given, a Karlik sheave unquestionably offers a lower rope pressure than an ordinary sheave.

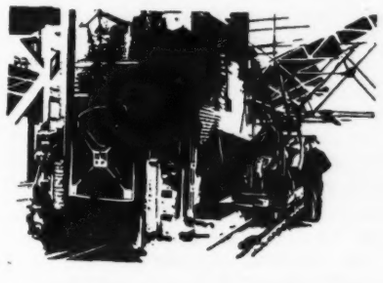
Another point in favor of the Karlik sheave is its high capacity. The figures in the example given should also illustrate its practical advantages. With a coefficient of friction between clamp and rope μ = 0.1 the capacity of the clamp carrying the heaviest load is $T - S = U = (2K + R) \times 0.1 = (3060 + 688) \times 0.1$ = 375 kg. This gives a drive factor $e^{K\mu\alpha} = e^{3.48 \times 0.1\alpha}$. For an arc of contact of 180 deg. the drive factor is 5.6. With complete utilization of the driving power the ratio of tension in both ropes may therefore be: $T : t = 5.6 : 1$. In this case the values are T = 10,800 kg. and t = 3,400 kg., $T : t = 3.18$. Safety from rope slippage is thus great.

A regular drive sheave with hardened lining has a coefficient of friction not exceeding 0.20; hence a driving factor 1.60. Even linings of Ferrodo fiber or similar materials not easily affected by oil, with coefficients as high as 0.35, do not give a driving factor above 3, showing that safety from slippage is not afforded. Such safety is, however, absolutely essential on account of the periods of accelerated and retarded speed, which cause great variations in rope tension.

The diagram in Fig. 3 shows the changes in the ratio $T : t = 3.18$ in the example given, when accelerating or retarding forces come into play. This ratio increases to 3.53, when the rope speed drops from 5 m./sec. to zero in 10 sec., or if, in this period of time, a speed of 5 m./sec. is attained. If by stronger braking action the retardation is doubled—that is, the hoist is brought to a stop in 5 sec., a distance of 12.5 m.—the ratio goes up to 3.94. The abrupt incline in the $T : t =$ curve by further shortening of the braking distance is worthy of attention. Although the Karlik sheave in this case possesses appreciable resistance to slippage, the limit for hoisting capacity would long have been passed with an ordinary sheave. Besides removing the danger from rope slippage, the Karlik sheave also permits a shortening of the starting and stopping period, thus increasing the hoisting capacity.

The fact must be considered, that with a load ratio $T : t = 1.6$, as given for wood-lined sheaves, the value t may exceed T when the brakes are applied on the heavier hoisted load. If, for instance, T', equal to 10,800 kg., is being hoisted and t', equal to 6,800 kg., is being lowered, stopping in 1 sec. with a rope speed of 5 m./sec. would give a ratio $t' : T' = 2.07$, or considerably above the hoisting capacity 1.6. The rope would consequently slip.

Many of the devices used with common sheaves to prevent rope slippage are made superfluous by the Karlik sheave, such as the use of an "under rope" much heavier than the "over rope," special brakes, double-groove reversible drive sheaves, and similar arrangements. Its introduction has resulted in an increase in the use of hoisting equipment of the Koepe type described and a higher degree of safety in the hoisting operations.



National Exploration's Unemployment Plan

A NEW FORM of protection for mine employees against unemployment has been evolved by the National Exploration Corporation, operating the Midnite Test mine, near Prescott, Ariz., according to A. L. Russell, of Prescott. During the latter part of 1931 the management decided to close down, pending financing and planning for extended operations. During this time several old-time employees had to be discharged, and failed to obtain employment from other operators. W. W. Linesba, the general manager, called the operating staff together and devised a plan that would insure protection in case of unemployment, protection from accidents while not on duty, and a chance to increase earning power.

The company plan keeps men employed if conditions permit; an unemployment and employee betterment fund is established and maintained jointly by the company and the employees, to be administered by three trustees, to compensate those men who are laid off from no fault of their own, to help those employees who are interested in learning more about their own particular work, to purchase a group policy against sickness and accidents to all members of the fund when off duty, and to provide recreational facilities. The plan also provides that records be kept of all employees, filed so that they may be available in connection with unemployment payments and for use by the employment departments of other mining companies.

The company intends to expedite all new construction work and development work on the Midnite shaft. Additional work on adjoining claims, not scheduled to start before September, 1932, will begin as soon as the snow is off the ground, permitting the erection of machinery. Additional housing facilities will be provided as soon as trucks can haul the material into the property. All work under the new plan will be figured on a basis of eight hours per man per day, no more and no less—no overtime for anyone. The idea is to put as much money into the hands of as many people as possible.

If curtailment of personnel is unavoidable, the men will be notified as soon as practicable by the general manager as to when the lay-off will take place and whether it will be permanent or temporary. A man discharged for cause will lose all rights under the proposed plan. Men laid off temporarily for disciplinary reasons will still retain their rights. A lay-off that affects the whole or part of the forces will allow payments to be made after a period of 60 days from the last pay day.

The second part of the plan calls for establishing a fund by deducting 5 per cent from the salary of each employee. This fund, to be placed in a regular depository at interest, will be administered by three men—the general manager, the employment official, and a representative of the men. A group coverage policy, which will protect the workers 24 hours a day against accidents and sickness, will be purchased from the fund, the cost to be divided between the fund and the company. An arrangement will be made whereby technical information will be available in the form of books, magazines, and regular study courses for those interested in getting ahead in their various departments. In addition to these benefits, recreational facilities will be provided. A plan has been suggested by one of the employees for a clubhouse, with billiard tables, bowling alleys, card rooms, dancing floor, and, later, a swimming pool. As far as can be learned, the management approves the plan.

Though payments from the fund may be necessary an account of unemployment, these payments will be made on a basis of length of service, number of dependents, previous salary, and service record. All payments will be based on 15 per cent of the previous average salary for one year, 20 per cent for two years' salary, and a 3 per cent allowance for each dependent after the second year. A maximum allowance of 30 per cent will be allowed. If the employee is already receiving state compensation, or payments from insurance carried by the fund, no payment will be made from the fund. No payments will be made after the employee has taken other work. Effort will be made to encourage all employees to make regular deposits in recognized savings funds and other investments, to protect them from the uncertainty of the future. One Prescott bank has already offered to establish an arrangement whereby deposits can be made at the offices of the mine.

A fourth part of the plan calls for the keeping of regular records of employees. Previously, the small number of men employed made this unnecessary. A card index will provide the man's name, address, age, physical characteristics, doctor's report, previous experience and references, and any other information the employment officer may care to add. A system of merits not unlike the railroad system is being worked out by Mr. Linesba and Mr.

Russell, to be put into effect about March 1. This system will allow each man 100 merits at the start of each half year. The manager will deduct such demerits as may seem to him to be right in each case. These demerits will not be used as a basis for discharging men, but only for the purpose of figuring payments from the fund. The keeping of the records for use with employment departments of other mines obviously requires no detailed explanation.

The general plan of National Exploration closely resembles Gerard Swope's General Electric plan, and was designed from this plan, but it meets several problems that General Electric will not have to solve. The character of the employees is different. Miners as a rule are wanderers and of a different disposition from any other class of workers. They do not take kindly to new ideas. They distrust banks and other institutions that encourage saving. The protection of their future will be a real job for National Exploration. Under present conditions the plan will work, and the hope is expressed that, by the time jobs become plentiful again, the management will have been able to educate the employees sufficiently so that continued operation of the plan will be successful.

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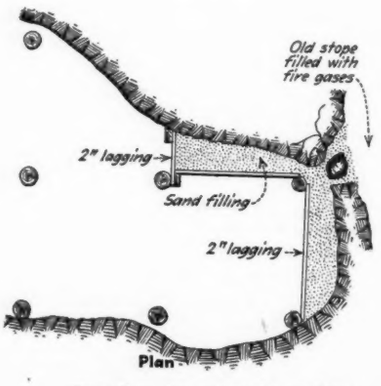
Cement-Gun Control Of Loose and Moving Ground

T. E. Smith

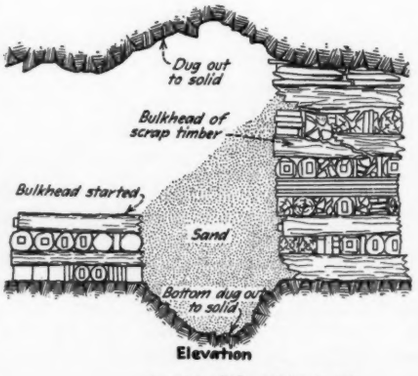
IN PLACES where the ground in a mine is moving rapidly, stoppings made of concrete crack in a few days, and allow bad air, heat, and noxious fire gases to leak into live workings. During efforts to control the Leonard fire at Butte, Mont., from 1928 to 1930, the Cement Gun crew was sent to all the gunited structures every two weeks to apply a new coat over the cracks and around the edges near the ground. A method developed in 1930 to use sand as a filling for dams and stoppings—where bad air, heat, or gas were to be held back in moving ground—has proved highly successful. It was first used to isolate, from the mine workings, abandoned areas in the upper levels of the Leonard mine.

A bulkhead is first built with old timber across the drift to be closed, as shown in an accompanying sketch. After this has been wedged tightly to the back and sides, the loose ground is taken from sides, back, and bottom. Another wooden bulkhead, started 6 ft. or more outside the first one, is brought up to about 3 or 4 ft. high. The space between the two is then filled with sand. If rails permit a car to be brought within shoveling distance, the sand can be delivered in mine cars and shoveled by hand. A water hose should be played on the sand continuously while it is being shoveled, to insure tight packing. The smaller grains are thus washed into the interstices between the larger grains; and, if the correct type of sand is used, it will pack so tightly that it would have to be removed by pick. If the drift has caved badly, and a mine car cannot reach the bulkhead, the sand can be shot into it with the Cement Gun. Sand shot through this machine is packed tightly. The water is mixed more uniformly, and a better job is done.

The outside bulkhead is raised as the work progresses, and the last sand is shot through a narrow opening at the top. This is filled with a block of wood after the sand is in place. When delivered by Cement Gun the sand will stand nearly vertical if 20 per cent or more of the material will pass a 30-mesh screen, and if the correct pressure and amount of water has been used. If the outside bulkhead is wedged tightly to the back and sides after the work is finished, it will withstand any applied pressure. Squeezing of the drift, caving of the back, or earth movements will tend to pack the sand more completely; a lateral movement of the drift that would crack a concrete structure is absorbed by movement of the grains of sand, one upon the other, without impairing the efficacy of the bulkhead as a seal.



Stopping fire gases in a stope



Sand bulkhead in drift

When delivering sand through a Cement Gun, variations of pressure, rate of feeding, or flow of water are not important. The operator should keep the nozzle moving steadily to and fro, and round and about, so that the sand may be placed in a homogeneous mass. If held in one place for a few seconds, the coarse particles rebound and fly several feet away, while the finer particles remain where shot. This results in a firm, dense mass in one place and a porous area around it.

A block of ore was being taken up alongside No. 1,663 raise of the Tramway mine, and on the second mining floor a shot broke into an old stope full of fire gases. The miners were taken out and the fire gang was sent in. The corner set that broke into the gas-filled stope had been opened on this floor, so its floor was solid. After the loose material was shoveled out, 2-in. lagging was placed outside the posts, leaving a space of several inches between them and the ground. This space was shot full of sand with a Cement Gun set on the sill 20 ft. below and about 50 ft. back from the manway; an accompanying sketch shows the arrangement. After the sand has been brought up nearly to the top of the timber, the back lagging of the corner set was removed, and the operator took his nozzle above the girts, and shot the

sand entirely up to the back. The opening into the old stope was near the top of the timber. After the back lagging was out, the operator was able to point the nozzle into the opening, and fill it. Nearly a mine car of sand was delivered into the hole before it was filled. Several large deep cracks were filled similarly. A pressure of 20 lb. was used. When the work was started, the gas was so dense that a man could work in it for only a few minutes. After the shooting was completed, no trace of gas come through the sand.

Lately, when retimbering a drift through loose, blocky ground, the Cement Gun was used to prevent the ground from running. This method was found to be more effective than timbering or any system of sprags and stulls. First, the side lagging was removed, and then the machine was used to distribute gunite over the exposed rock surface and into the cracks and interstices that could be reached. After three or four hours, this had hardened sufficiently to hold in place the loose, blocky ground while the back lagging was removed. While the gunite was hardening, the loose material on the track would be sent out. In a hot place the gunite would be fairly hard in two hours.

After removing the back lagging, or as much of it as was safe, and removing loose ground to make room for new timber, the nozzle man would climb up into the hole made and fill the cracks with a stiff mixture as far as he could reach, then give the exposed ground a coating about 1 in. thick.

For filling long and deep cracks, and holes in badly broken ground, a thin mixture would be used that would run into cracks that could not be reached directly. For filling cracks that can be reached, a thick mixture is better, for it gives considerable support to loose rocks, even before it has time to set. For coating the exposed ground, strong pressure and a relatively dry mixture are used. A considerable amount of gunite can be delivered into cracks over the set ahead, so as to steady the ground while the lagging is being removed.

The filling of gunite will steady ground and loose rocks better than a forest of sprags and stulls would while the timbers are being removed and new ones put in. Another advantage is that the least sign of movement will crack the gunite and thus give warning to the workmen under the moving ground. This method is particularly applicable to large blocky ground that cannot be worked with spiling.

In the job under discussion, about three mine cars of gunite material was required to steady one set. Each car would hold about 1,000 lb. of sand and three sacks of cement. Unless the ground is exceptionally heavy and broken, two men should change a set in two days and be perfectly safe all the time.

Lately, when sealing some of the fire country, opening of part of an old stope was necessary. The filling, largely blocky, would run on the slightest provocation. By putting in numerous sprags and stulls, the workmen could brace the loose ground back from the drift timbers as they rose. A gunite wall 10 in. thick was to be built between the drift timbers and the loose ground; consequently, all the sprags had to be removed, so that the forms could be built.

While the forest of sprags and stulls was still in place, a thick coat of gunite was applied to the entire exposed surface, and all the cracks and interstices between the rocks were filled with a stiff mixture. The area of exposed ground, 16x10 ft., stood almost vertical. After the gunite had hardened, the sprags were removed, and the forms built. Not a rock moved in the whole exposed ground. The slightest movement of any of the rocks would have been shown by cracks in the gunite coating. The sense of security felt by the miners was an added advantage.

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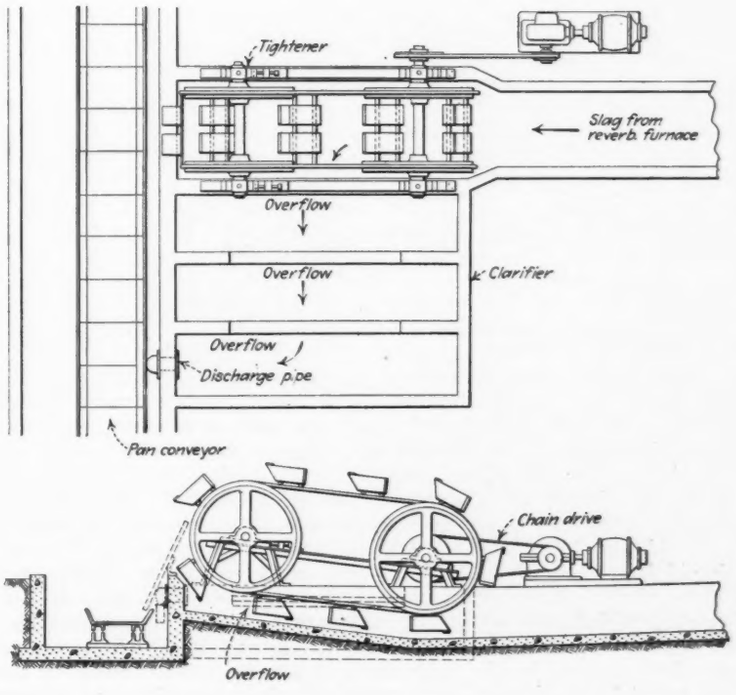
Novel Classifier Installation

CONSTANT TROUBLE was experienced at the smelter of Compagnie du Boleo, Santa Rosalia, Baja California, on the main pan conveyor used for transporting granulated reverberatory slag from the loading pit to the stockpiles. Rapid wear of the exposed conveyor parts, caused by the corrosive action of the sea water used in granulating the slag, necessitated frequent shutdowns and costly repairs. To overcome this difficulty a method was developed by Victor Mussio, smelter superintendent, whereby the slag was separated from the water and discharged on the conveyor in a semi-dry state.

As shown in the accompanying sketch, the arrangement comprises a classifier with perforated steel buckets, built after the principle of a drag classifier; and three concrete settling reservoirs, with overflow weirs. The coarse slag particles collected in the classifier pit are discharged over an incline apron on the conveyor, and clear water from the end reservoir flows through a drainpipe either to the pump sump or into the ocean. The classifier is actuated by a small motor-driven speed reducer, connected by a steel chain to the sprocket wheel shaft.



Open-Pit Iron Mining

Office and shops of Butler Bros., iron ore operators, Cooley, Minn., on the Mesabi Range

Details of operating practice and equipment in the mines of the Lake Superior region of Michigan, U.S.A.

A. H. Hubbell
Associate Editor

DIPPER TEETH and car wheels are matters of interest in connection with open-pit operation. Practice with respect to them is not uniform in the Iron Country even among neighboring pits. Manganese steel is widely used for teeth, but it has strong competition. One company is using plain forged steel, the tooth having two pieces, base and point. The company's buying practice is of interest, in that, customarily, it orders more than 250 teeth, so as to secure a discount granted on anything above that quantity. When the stock has dwindled to 75, it is replenished. These teeth are interchangeable and are divided among three pits having a total of four electric and two steam shovels. Their length of life is variable, depending as it does on the character of the digging, which in turn depends to some extent on whether or not the ground is shaken up by blasting ahead of the shovel. Fifteen to thirty minutes is required to change a set of points. An extra set is always carried on the "stone boat" (a sled-like affair that is sometimes dragged behind the shovel). In contrast to this latter point of practice, another company runs a set of points down to the shovel from the storehouse, when needed, using a "gas speeder" or gasoline-driven section car for the purpose.

Another company forges the teeth required for its steam shovels, but on its newer electric shovels uses teeth with a plain forged-steel base and manganese-steel points. These are said to handle 10,000 to 20,000 tons, on an average, lasting approximately two 10-hr. shifts in ore, in ordinary digging, half that time in difficult ground, and about one week in stripping. Two sets are kept on hand for every shovel, gas speeders being used to deliver them, if necessary. Two other companies forge their teeth from mild-steel billets and face them with old locomotive tires. When one of these teeth becomes worn, a piece of tire is welded to it and sharpened.

In two other pits, on different ranges but operated by the same company, molybdenum-chrome steel teeth (Westeco Molychrome) are used. These are credited by the users with long life, in fair digging. They are reversible and can be resharpened.

From this brief and limited survey of the practice at approximately twenty pits one can only conclude that other factors also besides the wearing quality of the material of which dipper teeth are made influence their selection.

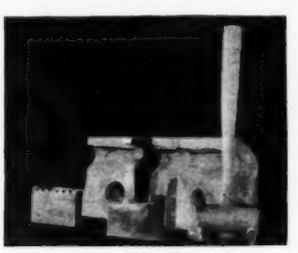
So variable is the practice likewise in regard to car wheels that one is forced to a similar conclusion. Fewer materials are involved. Car wheels are usually of cast iron, with a chilled tread, or of steel, the latter being either rolled or cast. Manganese steel is being used to some extent.

Cast-iron wheels are widely used, despite certain disadvantages. Some users claim they wear out too rapidly, even with a chilled tread, and that they develop flat spots. These are held responsible, by one user at least, for breakage of arch bars on the cars. Another user holds that the resultant frequent changing they require adds to the expense, so that there may be little difference between their final cost and that of cast-steel wheels. But repurchase is invited and facilitated by the fact that on new purchases some wheel manufacturers will accept and credit the old wheels at their scrap value. In one case, the old wheels are sent, axles and all, to the foundry, where they are pressed off and replaced by new wheels for the customer. This is necessary where the mine shop has no wheel press. Moreover, one operator who has tried cast-steel wheels asserts their extra life is insufficient to compensate for their extra cost as compared with cast iron. Experience with wheels of either material varies, performance depending to some extent on circumstances.

The so-called Davis wheel, with manganese in flange and tread, is used at a certain pit, but in two other mines of the same company, a light "one-turn" A.R.A. standard wheel of plain cast-steel is in service, demonstrating the absence of fixed practice. Another operator is using the Carnegie rolled-steel wheel—of ordinary steel—and when the flanges become worn, builds them up with the welding torch. The tread of this wheel can be turned twice. When worn out, the wheels are returned as scrap. Another company also is using a rolled mild-steel wheel, resembling a cast-iron wheel in section. These, when worn, are turned in for credit on new wheels. They are not built up with the torch or otherwise repaired. According to the superintendent at this property, a 33-in. wheel of this type weighs about 550 lb. and cost them $17, in contrast with a cast-iron wheel of the same diameter which weighs 650 lb. and cost the company $13.

Use of the welding torch for repairing wheels is surprisingly limited. Some of the mine shops visited use it for repairing flanges, apparently to their satisfaction. Others evidently have not come to regard it as worth while for this application.

Few wrecks appear to be caused by wheel failure. An occasional derail-



Slotted brake shoe, jigs and copper hammer, used in turning down locomotive tires at Butler Bros. plant, Cooley, Minn.

ment from a broken flange seems to be the nearest approach to a wreck. The reduced sped at which pit haulage is conducted doubtless accounts for this.

Locomotive tires being in somewhat the same category as car wheels, space will be taken here to describe the method recently adopted by Butler Bros., iron-ore operators, in their shops at Cooley, Minn., on the Mesabi, for turning down locomotive tires without removing them from the locomotive. The method was developed by the Great Northern railroad some five or six years ago, but it is probably not widely known. Details are presented through the courtesy of Mr. R. O. Hocking, manager of mines. According to the master mechanic, Mr. Gust Weggum, at Cooley, the procedure is as follows:

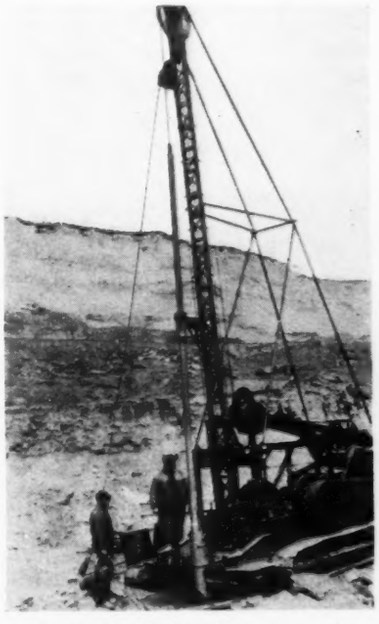
The brake shoes are slackened off and removed. In their place are put other brake shoes that are the same as those removed, save that they have been slotted with two slots 1 in. wide and 1 in. deep. The slots hold special jigs. Slots and jigs are shown in the accompanying photographic reproduction. The jig used for cutting the tread has six ¼-in. grooves, ¼ in. deep and ¼ in. apart, with a quarter-inch shoulder of metal at each end. Another jig is shaped for cutting the flange, and a third for facing the outside edge of the tire.

Drilling ahead of the shovel in a Mesabi pit. A caterpillar-mounted Armstrong deep-well type blast hole drill is putting down 6-in. holes to an average depth of 33 ft.

These jigs are fitted into the slots and adjusted with soft copper hammers, so as to fit in the proper place. The tread is turned first. One of these special brake shoes with its jig is put on every wheel. Each brake shoe is then carefully readjusted (because it has been slackened off in the first place to get it off).

Some sort of a hoist that will pull very slowly and uniformly has in the meantime been rigged up. Actually, an old steam shovel is used for pulling the locomotive on which the tires are to be turned. The shovel is coupled to the front end of the locomotive. From 600 to 800 ft. of cable is used, one end being fastened to the shovel hoisting drum and the other end to the track at a point 600 to 800 ft. distant. Steam is piped from the locomotive to the throttle on the shovel. The locomotive, with its brakes set and with air applied to driver brake cylinders at suitable pressure (generally 10 to 15 lb. per square inch) to feed the cutting tools into the tire, is now pulled slowly, perhaps only 2 miles an hour, for a distance of about 800 ft. Then the brakes are slackened and the locomotive is returned to its starting place under its own power. The jigs, each in its respective brake-shoe slot, are now adjusted with the copper hammers so that the cutting edges are in position to cut the metal that was previously between the spaced cutting edges. Then the locomotive is pulled again over the 800-ft. stretch and again allowed to return. This is kept up, the jig being shifted each time as described, until the tire has been turned down sufficiently. Then the flange-cutting jig is substituted for the tread-cutting tool, and a similar procedure followed, save that there is no need to shift this jig each time. Then the third jig is used to face the outside edge in similar fashion. The jigs must be kept tight against the face being cut, even if the brake rods have to be tightened up.

Length of time required depends on the condition and hardness of the tire. Chilled spots may have to be ground down with an emery wheel. All wheels are done at the same time. To turn the wheels down in this manner requires 8 to 18 hr. Doing the job the old way required the removal and replacement of the tires, taking four men one day. Then the tires had to be transported to another company's shops for turning, because the local shop lacked the equipment. Four men are required by the new method also, but it is cheaper in time and money.



Deep-hole drilling in an open pit at Marble, Minn.

Blasting methods employed in the Volunteer and Tilden pits, both on the Marquette Range, have been described briefly in earlier issues. In both instances the use of drill, or well, holes was involved, in contrast with tunnel blasting. Blast hole drilling in single rows is also commonly employed on the Mesabi in breaking ground ahead of the shovels. The practice varies to some extent and in general is rather different today from that formerly employed.

At certain Mesabi Range pits not previously referred to the practice may be taken as typical of the modern way. The holes at these pits were formerly drilled with an I.-R. D113 machine, which has a strong upstroke. They were 26 ft. deep, were started at 2⅞ in. and bottomed at 1 in. They were spaced about 15 ft. apart and staggered in alternate rows. Chambering was used, the result being that many large pieces of rock had to be handled or broken with pop shots. This cut down the shovel tonnage and increased the cost of shovel maintenance and operation.

Today, holes 6 in. in diameter are put down to an average depth of 40 ft., being bottomed normally about 6 ft. below the working grade. Drilling is done with a so-called well-type blast-hole drill made by the Armstrong Manufacturing Company, Waterloo, Iowa. This drill is electric-driven and crawler-mounted and is consequently exceedingly mobile. The holes are column-loaded with 60 and 80 per cent Hercules Gelamite, the proportion of the two depending on the ground. They are loaded for approximately half their depth, and as many as 100 holes are detonated at a time, the detonator used being No. 6 Cordeau Bickford plus a single cap. The spacing of the holes laterally and their distance from the bank depends on its height and nature —something to be determined by experience. Where the number of holes is smaller, electric blasting caps may be substituted for the Cordeau.

In loading holes, a length of Cordeau is hung in each hole reaching to the

A battery of drills working in same pit as that shown in other photographic reproduction on this page

bottom, the detonator thus traversing the entire charge. The outer free ends of these individual lengths are then spliced to another length of Cordeau lying on the surface parallel with the row of holes and thus tying them all together. If the blast includes more than one row, the Cordeau on the surface runs midway between the first two adjacent rows, thence doubling back and running between the next two rows, and so on until the Cordeau in every hole in the blast is tied into the main line, making a single circuit and permitting simultaneous shooting. A single fulminate cap on one end of the surface length of Cordeau is sufficient to detonate the entire charge, the detonating wave traversing the Cordeau at such a speed that all holes are fired seemingly at the same time, although one hole actually follows another.

The joints or splices are made by splitting the top end of the length of Cordeau in each hole, and wrapping the two ends thus produced around the connecting length. The unsplit portion must be held tightly against the latter. The cap also must be held tightly against it. Cordeau is lead-covered and may be countered with cotton for additional protection to the lead. It must not kink; otherwise it may break and misfire. It is also being countered with brass wire to prevent kinking and is curved gently over the casing, as it goes down the drill hole, and not bent sharply. Second-hand pipe is used for casing and is allowed to project enough at the top of the hole to prevent rocks from falling down. Cordeau is not affected by water. Missed holes have always been serious in shovel work, but are less serious when this detonator is used. The shovel can work up close with much less risk of detonating the Cordeau than if a blasting cap were there.

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Handling Diatomite From Quarry to Silo

L. J. Walters

DIATOMITE mined by the Dicalite Company, at Palos Verdes Estates, near Wilmington, Calif., comes from underground and open-cut operations. In the latter, a dragline scraper removes the overburden, dumping it into a glory hole, with a bin bottom, equipped with a Moore gate. A train of four or five 4-yd. side-dump cars is pushed into a tunnel under the bin bottom, where they are loaded, taken out to the waste dump, and discharged. After the quarry has been stripped of overburden, the crude ore is broken from the face by blasting, and rolled down to the track of a dragline scraper that travels across the face of the quarry, picking up the material deposited in its path. This material is delivered into a pocket, from which cars are loaded, to be delivered to the receiving hopper at the plant, shown in Fig. 1.

The top of the receiving hopper, which has a capacity of twelve cars, is about flush with the track, permitting the cars to be easily and quickly dumped, avoiding the danger of a car body fouling the bin and being tipped into it when the train makes its next forward move. The hopper is equipped with a 36-in. Stephens-Adamson steel-pan conveyor, set at an angle of 24 deg. to the horizontal and traveling 10 ft. per minute. Depth of feed is controlled by a double rack-and-pinion gate, manually operated from the front face of the bin.

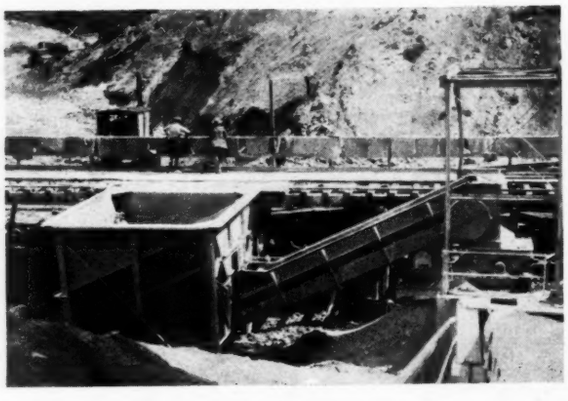

Fig. 1—Receiving hopper with pan conveyor. Note covering on conveyor in foreground.

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Average size of lump is about 24x18 in. No trouble has been experienced from the excessive slope of the conveyor. The material rides well and does not roll backward down the incline. The angle of repose and frictional properties of diatomite is such that all bins and chutes have to be designed with a minimum angle of 55 deg. This, of course, absorbs much valuable headroom.

A Pennsylvania crusher is used to break up the crude material. It is of the swing-hammer type, with hopper opening of 32x24 in.; is equipped with a tramp-iron throw-out feature; and is direct-connected through flexible coupling to a 50-hp. motor. Crushed material, about 20 mesh, with 30 to 50 per cent moisture, falls onto a 24-in, Goodrich belt conveyor with S. A. carriers, traveling 200 ft. per minute, 22 deg. to the horizontal. The product is conveyed easily, has little dust at this stage, does not roll readily, is compact, and stays on the belt as loaded. However, it is highly abrasive. Gearing and bearings must be protected. Diatomite also causes leather belts to deteriorate

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Fig. 2 — Complete plant; loading track and quarry in foreground. Note tail sprocket of pan conveyor under silo.

quickly, but does not appear to be so injurious to rubber.

The conveyor belt underneath the crusher discharges onto a short cross belt of the same width and speed, which in turn discharges onto another 24-in. belt, which elevates it to the top of No. 1 bin, where it goes over a fixed tripper, loading this bin from both sides of the belt; or the feed can be bypassed from No. 1 bin and taken to No. 2 bin, where it can be fed to either side of the bin, in any quantity desired. This system permits supplying the bins with a varying amount of material, as desired, and allows blending any two products from different parts of the deposit. The diatomite structure varies in different quarries in the same deposit, and a blend is invariably desirable.

The two silos shown herewith have a combined capacity sufficient for ten hours' plant operation, thereby avoiding night-shift work later than midnight in mine or quarry. They have a false bottom at an angle of 60 deg., feeding to a central longitudinal opening, under which steel apron feeders carry the material to a common point between the bins, and discharge it onto a 14-in. belt traveling at 350 ft. per minute, which takes the material to the processing plant. These steel apron feeders, of Stephens-Adamson design, are of specially heavy construction. They must support a column of material 55 ft. high when the silos are fully loaded. They are designed to give a live bottom to the silos, and must empty them under all conditions of load.

So far, no arching has occurred, but poke holes have been provided in the concrete walls which will permit of breaking an arch down should it form. Diatomite arches readily, and all precautions for the prevention of such conditions have to be taken. The feeders, 30 in. wide, usually travel at 10 ft. per minute, and depth of feed is controlled by slide gates, equipped with a minimum opening device, preventing the operator from closing the gate too close to the feeder. When operating under high heads these feeders are almost positive in action. Should the discharge opening be constricted too much to permit most of the material being moved to pass freely, a wreck would result.

Each feeder has its own drive, equipped with a variable-speed control in addition to the speed variation obtainable by changing sprockets on the chain drive from motor to speed reducer. The apron feeders can thus be operated to meet any and all requirements for blending or production volume. Each belt conveyor is driven from one countershaft, parallel to the short cross conveyor. The motor on this drive is electrically interlocked with the drive on the 36-in. pan conveyor ahead of the crusher. Should a sudden stoppage occur beyond the crusher, all the feed is automatically shut off from the system.

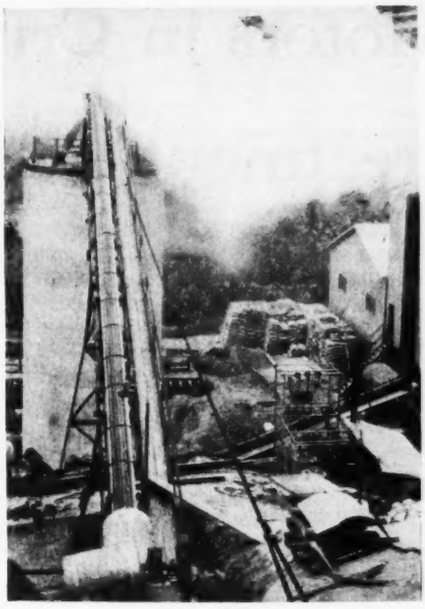

Fig. 3—Conveyor to silos, with cover framework

The circular steel-plate covering for the conveyors, shown in the illustration, was adopted to secure minimum first cost of installation. Savings effected over the usual type of bridge construction were appreciable, and the innovation justified itself in every way. The only disadvantage is that the operator's walkway is exposed to the weather. In California this is not a serious consideration. The circular covering has been found to be adequate against the heaviest winds, is rain-proof, and affords ample protection from the sun.

Lubrication connections are piped to convenient points on the walkway, and openings are provided at each carrier, to permit of easy repairs or renewals when required. Electric wiring is in conduit, and motors are fully protected with the necessary relays. Start-and-stop buttons and emergency stops have been placed at convenient points. The advantage of a bucket installation was given consideration, but the factors of abrasion, inaccessibility, and maintenance cost influenced decision in favor of the belt conveyors, with excellent results.

The cost per ton of the material handled has been satisfactory, and the equipment, barring unforeseen accidents, will probably outlive the remainder of the plant.

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Sampling Sand and Gravel

FOR preliminary prospecting of the Clowdy deposit of sand and gravel, near Dallas, Tex., holes 300 ft. apart were sunk to water level by using a 6-in. auger with an extension handle made of ¾-in. pipe, according to R. L. Windrow (I. C. 6581, U. S. Bureau of Mines). Holes up to 20 ft. in depth can be sunk with these augers if water or very sandy material is not encountered. The cost is about 15c. per foot. This type of testing was used only to determine the boundaries of the gravel deposit and also the depth of the overburden.

After the gravel deposits were outlined by means of augers, test pits spaced 600 ft. apart were excavated to determine the thickness and quality of the gravel. Some of these pits were dug by hand and some were excavated with a 1-yd. caterpillar-type gasoline dragline crane. The crane would have been used to dig all the pits except for the fact that part of the property is about 2 miles west of the plant, and the time lost in moving the crane to this part of the deposit would have offset the additional expense of digging the holes by hand. Test pits of this type cost about $1 per foot of depth when a machine was used, and $1.50 per foot when dug by hand.

Attempts were made to determine the thickness of the gravel on part of the Clowdy property by using a 6-in. well casing and a 5-in. slush bucket to sink through the gravel under water. This method was not satisfactory where the material contained an appreciable amount of gravel larger than 1½ in., because of slow progress in sinking.

After the areas and thicknesses of the gravel deposits were determined, the volume of gravel was calculated and converted into tonnage by assuming a weight of 3,000 lb. per cubic yard. Screen analyses of the material from various test holes were then made, and the average percentage of sand and gravel that should be recovered from the property was estimated.

Two methods were used to obtain samples for screen analysis. When test holes were dug with the dragline, the overburden was cast on one side of the hole, and the gravel on the opposite side; then an average sample was selected and shoveled into a sack. When test holes were dug by hand, samples were obtained above the water line by removing two shovelfuls of material from the side of the test hole. These samples were taken at regular depth intervals of 3 ft., and all samples from a hole were mixed to obtain the average of the material above the water line. Samples from below the water line were removed from the center of the holes at intervals of the same depth.

Electric Motors in Crushing and Treatment Plants

R. S. Sage
Industrial Engineering Department, General Electric Company, Schenectady, N. Y.

ELECTRICITY is the principal form of power applied to the machines used in ore preparation and treatment. The single motor formerly used for nearly the entire plant, or for a complete section, has given way to almost complete individual drive. Most large plants are divided into duplicate and parallel sections, to facilitate adaptation to varying output conditions and to permit servicing or making repairs without reduction of output.

An ore concentrator requires many motor drives. Some large mills operating on low-grade copper ores have installed as many as 1,500 to 2,000 motors, ranging in capacity from a fraction of a horsepower to as high as 500 to 600 hp., and occasionally even larger. As the process is a uniform and continuous one, an adequate and reliable service is imperative. The great variety in character and size of machines necessitates the use of driving motors of various types, to meet best particular operating conditions. Care is therefore desirable in the selection of motors, with respect to their suitability for torque requirements during starting, with reasonable demands upon the power supply, and for high efficiency when running at full speed during normal operation. This selection is also influenced by other considerations such as local conditions of temperature, moisture, dirt, fumes, and by the form of mechanical transmission employed—belting, gearing, or coupling. Equally important is the proper selection of starting and controlling equipment for the motors. Reliability and simplicity in operation and construction are essential.

The adoption of magnetic-type starters, with automatic switching wherever possible, is recommended. The personal factor and guesswork are largely eliminated, and equipment is started properly under all conditions.

Most machinery in this service is driven by alternating-current motors, the usual exceptions being direct-current motors for "mule" hauls, for up-grade approaches to the crusher, for cranes, and occasionally for variable-speed feeders and conveyors.

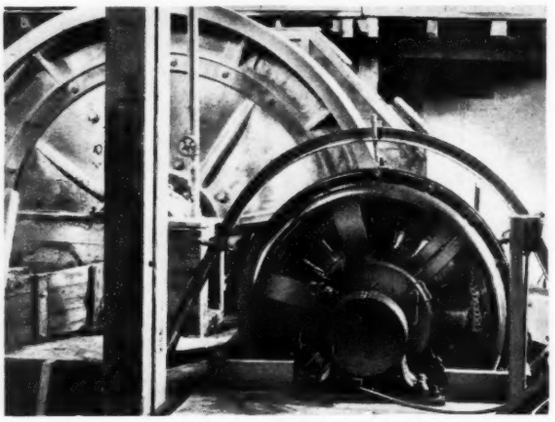

Fig. 1 — Marcy mill driven by a 150-hp., 360-r.p.m. direct-coupled super-synchronous motor

For hauling railway-size ore cars up an incline to the coarse crusher, direct-current motors have been found most satisfactory. With this type, extremely slow speeds can be obtained for moving the "mule" to engage the car, and automatic "slow-down" is possible by dynamic braking. For some of the most recent installations where high car speeds are used, the Ward Leonard control system has been employed with greater precision in handling, with higher operating efficiency—especially on continuous operating cycles — and somewhat greater safety. "Mule" haul motors for the most part are of the heavy mill type, usually of 150 hp. capacity. This equipment is generally installed in a separate building, and is not subjected to dust to any extent.

For the operation of rotary car dumpers, slip-ring induction motors have proved satisfactory, the control being of the simple drum-switch and resistor type. These motors, being mounted upon the dumper, are subjected to dust-laden air and should be protected.

Primary crushing is accomplished by jaw or gyratory crushers, which require up to 500 hp. for driving. For the former, the slip-ring motor has found most frequent adoption. This crusher is usually equipped with heavy flywheels, which, together with the liability of occasional starting while jammed with rock, render the squirrel-cage type less suitable by comparison. Connection usually is by rope or belting.

The gyratory crusher also usually employs the slip-ring driving motor, although the squirrel-cage type also has been used. In this case, normally, the starting is made with the crusher empty. If a stoppage occurs with rock in the crusher, it must be relieved before restarting.

In both of these applications, open-type motors are used, especial care being taken in some installations to exclude dust from the bearings. Both manual and magnetic-type starters are applicable, full-voltage starting being satisfactory for the squirrel-cage motors if line conditions permit.

Secondary crushers usually include the gyratory, roll, Symons cone, and Symons disk types. Rolls usually employ one slip-ring motor belted to each of the two rolls, although squirrel-cage motors have been used in a similar manner. Capacities range up to 150 hp. Drum switches, with resistors for the former and reduced voltage or full volt-

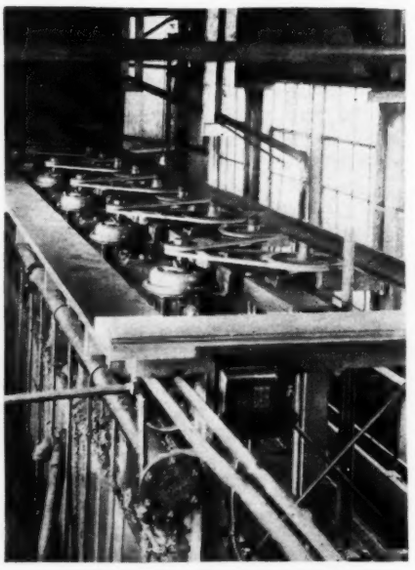

Fig. 2—Inverted motors, rated 3 hp., 1,200 r.p.m., drive sub-aeration flotation machine impellers through Texrope connection

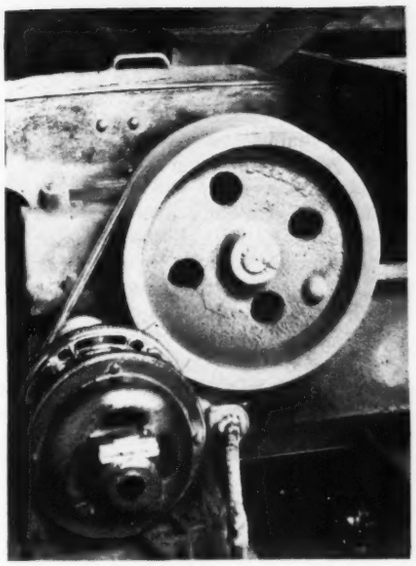

Fig. 3—Wilfley table driven by 1½ hp., 900-r.p.m. induction motor

age starters for the latter, are ordinarily used. Under most conditions the motors should be protected from falling material, because of their proximity to the rolls or conveyors that serve them. Symons disk crushers are generally driven by belted squirrel-cage motors with reduced voltage starters. The Symons cone crusher is motored with belted or direct-connected drives. Squirrel-cage, slip-ring, or synchronous motors may be successfully applied to this crusher, a selection in a given case in general depending upon the size of the crusher, the importance of limiting the load demands upon the power lines during starting, and on power-factor correction. Unloading this type of crusher, completely or partly, before starting is necessary. The driving power requirements range from 25 to 300 hp., for crusher speeds of 430 to 575 r.p.m. From the standpoint of simplicity, the squirrel - cage motor stands first, although superior starting conditions are obtainable with the slip-ring type. The use of the synchronous motor is indicated solely by its ability to correct power factor. Starting equipment may be any of the standardized manual or automatic types.

Ball and rod mills require torques considerably in excess of the normal running value, and in prevailing practice the starting torque of the motor is established at about 175 per cent of that at full load. After the mill has made an incomplete revolution, the required torque drops to approximately full load, as the mill inertia and the accelerating power requirements are relatively low. In contrast with coarse-crushing machinery, the power required by grinding mills is nearly uniform and continuous; furthermore, fine grinding is usually the most costly of any of the milling processes; and, for this reason, especial care should be taken in the selection of driving mechanisms.

Mills of the smaller sizes are usually belt-connected to the motors. With the larger ones, gear units are used, or the mill pinion shaft is directly coupled to the driving motor. The latter arrangement is preferable from the standpoint of simplicity and reliability, but is practicable only in applications where the speed of the mill-opinion shaft is not unduly low. Slip-ring motors are now rarely applied in this service; usually they are used only where limitations in power supply prohibit the use of squirrel-cage or synchronous motors. The squirrel-cage motor has generally been employed, but the present tendency is toward the synchronous type.

The squirrel-cage unit is chiefly applicable where reduction gears and a motor of moderate or high speed can be used, with resulting low first cost and reasonably high power factor and efficiency. The simplicity and sturdiness of this type of motor are strong factors in its favor. Usually, the high starting torque required necessitates starting at full voltage, and the double squirrel-cage rotor design may be desirable to obviate excessive starting currents.

As in many other fields of electric-motor drive, the principal advantage of driving grinding mills by synchronous motors is their ability to operate at high power factor and to raise the power factor of the power lines to which they are connected. Efficiency also is a consideration; and for the service under discussion the ability to build such motors at the slow speeds necessary for direct coupling to the mill pinion shaft, thereby avoiding intermediate reduction gears, accounts in large measure for their adoption. The necessary starting characteristics can be obtained in the simple synchronous motor by the proper design of starting winding, combined, if occasion requires, with the use of a somewhat oversize motor. As with the squirrel-cage motors, starting on full voltage is usually necessary. Where exceptionally low starting currents are imperative, no simple synchronous motor can be used; but these currents can be obtained by several special motors now available, including the so-called super-synchronous, the synchronous-induction, and certain other special forms.

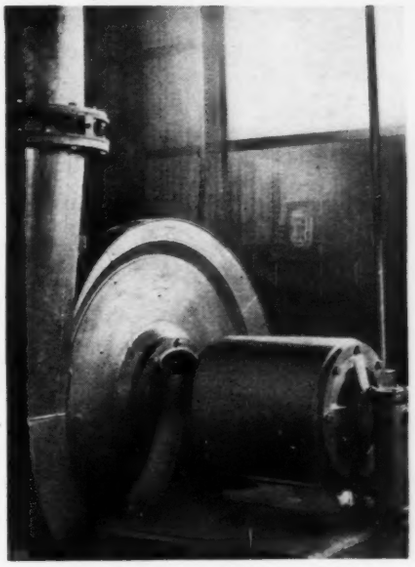

Fig. 4—Flotation air is supplied by these 65,000 c.f.m. 2-lb. centrifugal compressors, driven by 85-hp., 3,600-r.p.m. squirrel-cage motors

Ball mills may require up to 600 hp. with pinion-shaft speeds ranging from 150 to 450 r.p.m., depending on the size of the mill and type of gearing. Only rarely are these mills operated at variable speeds.

Motor capacities for belt conveyors range up to 250 hp. for belt or gear connection. With inclined conveyors, the motors usually are provided with solenoid brakes for holding a loaded belt during a shutdown. The most suitable motor for this service is the high-torque, squirrel-cage type, usually designed with double bars in the rotor. Starting torque specification range from 125 to 175 per cent of full load, and occasionally as much as 250 per cent is considered advisable. Manual or magnetic-type starters may be used. Although the full-voltage type is preferred, it may be desirable, in order to minimize initial shock, to use the reduced-voltage type, the starter utilizing a resistance in the primary circuit being the most suitable. Even though the conveyor may not start with this resistance in circuit, tension is put on belting and gearing. When

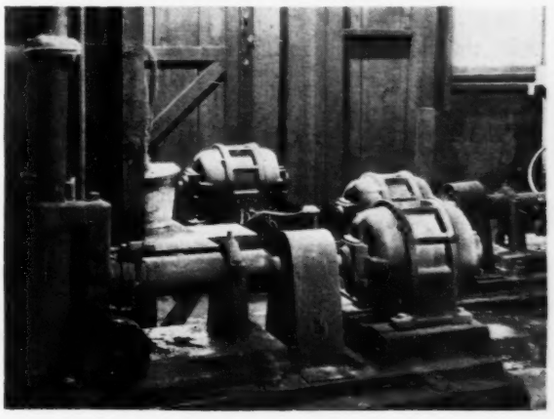

Fig. 5—Wilfley pumps with 20-hp., 1,800-r.p.m. squirrel-cage motors

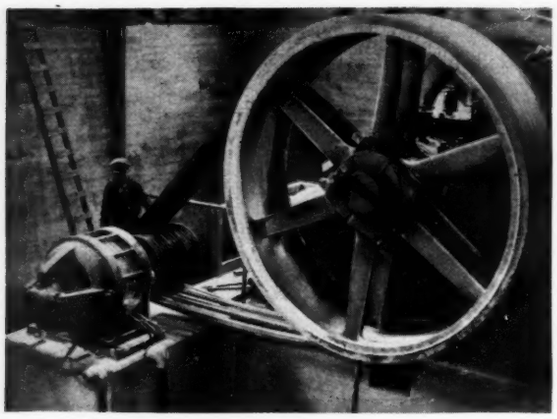

Fig. 6—200-hp., 600-r.p.m. slip-ring induction motor, connected by V-belt drive to Buchanan jaw crusher

the resistor is short-circuited, applying full voltage to the motor, the mechanical equipment is appreciably relieved of jar and shock. The auto-transformer type of starter also is applicable. Slip-ring motors may be employed successfully in this service, and their use may be justified in small plants or where step-by-step starting is desirable.

Belt conveyors serving secondary crushers or grinders have been driven by variable-speed, direct-current motors, in order that rate of feed might be controlled. Such motors usually are designed for a 2:1 speed range for constant torque, speed variation being obtained by adjusting the field current.

The practice of interlocking between main and cross-feed conveyors, and between conveyors and their associated crushers, is prevalent and usually easy of accomplishment. By this means the starting and stopping of the various units in the proper sequence prevents the accumulation of material at points in the system when starting the mill, or during an emergency shutdown of individual units.

Apron conveyors, feeders, and bucket elevators likewise are satisfactorily powered by squirrel-cage induction motors, the sizes and speeds involved usually being such as to render suitable those with standard-torque characteristics. Occasionally, variable-speed operation is desired, in which circumstance a choice between direct-current motors, brush-shifting a.c. motors, slip-ring induction motors or mechanical speed changers is available, depending on the size of the motor involved.

Shaking tables are generally equipped with individual driving motors, although group drives are still to be noted in a few instances. Individual drive is usually by means of 1½-hp., 1,200-r.p.m., squirrel-cage motors, capable of developing at least 150 per cent starting torque and preferably more, so as to permit starting the tables under the most adverse conditions. V-belt connected motors also are sometimes used in this service. Across-the-line, three-pole contactors, with pushbutton control or small manually operated oil switches—both providing protection against overloading—give suitable means of starting.

Flotation machines that require power for the mechanical agitation of the pulp may utilize individual drive or employ a single motor for a group of impellers, such as in M.S. machines. For the latter, squirrel-cage or slip-ring motors are used, connected by flat-belt, gear, or V-belt, or directly to the horizontal drive shaft. The motor capacity may range up to 150 hp., depending on the number of cells in the section constituting a group. The motor should be able to develop at least full-load starting torque. Janney machines, individually motored, and requiring from 6½ to 12 hp. per cell, and special, vertical, 600-r.p.m. motors are used, the impellers being mounted on the downward extended motor shaft. A cast-iron plate beneath the motor protects it from splashings from the cell below. On other types, such as the Fahrenwald and the M.S. sub-aeration type, 5-hp. open-type motors, mounted vertically, drive one or two impellers through V-belts.

Standard manual or automatic starters are applicable in this service. In M.S. machines with group drive, cold grease in the bevel gearing or partial blocking of the impellers by pulp may occasionally necessitate the application of full voltage to squirrel-cage motors before the machine will rotate.

Compressed air for flotation cells is required at pressures ranging from 2 to 5 lb. per square inch, depending on the type of machine. In this service are employed both positive-pressure and centrifugal blowers. Squirrel-cage motors, and occasionally synchronous motors in the larger sizes, supply the driving power for the former type, to which they usually are short-belt connected. Centrifugal blowers used for this service are usually of the single-stage type, with direct-connected motors forming a two-unit set, the smaller motors being of the squirrel-cage and the larger ones of the synchronous type. A noteworthy example of the latter are the 90,000-c.f.m. units at the Nevada Consolidated plant. These machines supply air at 2.5 lb. per square inch, and are driven by direct-connected, 1,250-hp., 3,600-r.p.m. synchronous motors. Starters for this service are standard, partial-voltage, manual or automatic voltage.

Water pumps and sand pumps for the most part are of the centrifugal type, for which the squirrel-cage motor is admirably suited. For the higher speeds, the motors usually are direct coupled to the impeller shafts; in other installations belting is used. Motors and starters of standard construction and insulation are applicable, although in some exceptionally wet locations drip covers over the motors are advisable. In some mills, return-water or other pumps are of the reciprocating type, with geared motors. Synchronous motors are entirely applicable in this service, especially in the larger sizes above about 100 hp., and in some instances have been so used, with resulting advantage in efficiency and power factor. Unless the motors are designed for especially high starting torque, bypassing will be necessary with plunger-type pumps during starting.

Other ore-milling apparatus to which squirrel-cage motors also find successful application are concentrate filter presses, jigs, classifiers, trommels and other screens, vacuum and air compressors, and sampling machines.

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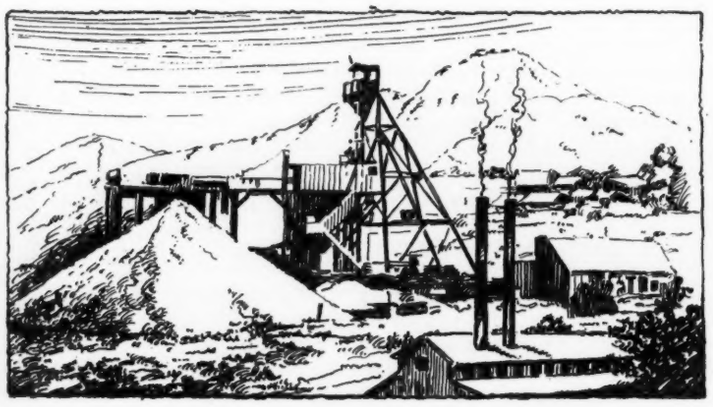


Cost of Square-Set Mining

Morris J. Elsing

AS A STOPE ENGINEER for the Copper Queen mine, at Bisbee, Ariz., in 1907, I kept rather complete personal notes of all matters connected with my work. In 1907, practically all ore was being produced from square-set stopes, and I noted that 8,950,000 cu.ft. of timber was used in these stopes and 475,000 pieces of 5-ft. lagging. This timber was used in 31,400 square-sets, and 565,600 tons of ore was produced. From these figures are calculated the results given in Table I.

Table I—Timber in Square-Setting, Copper Queen Mine

Board feet timber per set	285
Board feet lagging per set	15
Total board feet per set	300
Board feet timber per ton	15.8
Board feet lagging per ton	0.8
Total	16.6
Tons of ore per set	18.0

These figures are accurate, aside from certain qualifications, to be explained later, which have to do with the tonnage produced per set.

In square-set mining the usual practice is to frame timbers so that posts are 5 ft. center to center, and the height of the set usually varies between 7 and 8 ft. for stope sets and one foot higher for the sill floor sets. Square-sets at the Argonaut contain 209 cu.ft. and at the United Verde Extension 245 cu.ft., with posts 5 ft. 6 in. center to center. At the Park Utah sets contain 183 cu.ft.

The amount of timber needed per set varies slightly with the design of framing. The calculations in Table II represent the average amount of timber ussed with sets 5x5 ft. and posts 7 ft. 2 in. long.

The set for which details are given in Table II represents 200 cu.ft. of ground and with an ore having a specific gravity of 12.5 cu.ft. per ton, 16 tons will be produced per set. On this basis the board feet per ton of ore produced will be 8.8, 12.2, and 16.4 respectively when using 8x8 in., 10x10 in., and 12x12 in. timber. On the assumption that this framed timber costs 4c. per board foot, the cost per ton for timber is 35c., 49c., and 64c.

Table II—Timber in Square-Sets, 5x5 Ft., Posts 7 Ft. 2 In.

	Board Feet Per Set 8x8 In.	10x10 In.	12x12 In.
1.2 posts (a)	51	80	115
1.1 caps	27	42	59
1.1 ties (5x8 in.—6x12 in.)	19	21	30
Stope lagging	18	18	18
Chute lining	10	10	10
Blocking	7	11	16
Flooring	2	2	2
Miscellaneous, wedges, breakage	9	11	13
Totals	143	195	263

(a) A single square-set requires four posts. Two square-sets together require six posts, or three posts per set. The larger the section the fewer the posts. If a second section abuts the first the second requires fewer posts. I have estimated an average of 1.2 posts per set. Similarly with caps and ties.

In most mines where square-sets are used, the specific gravity of the ore varies widely, the range being from 8.5 cu.ft. in place per ton for heavy sulphide to 15 cu.ft. per ton for light, oxidized ore. The figure of 12.5 cu.ft. per ton is probably close to the average. In all square-set mines some overbreakage is obtained, which gives a greater production per set than is accounted for by the volume of the set. The overbreakage probably varies from 5 to 10 per cent and reduces the amount of timber that is actually used in the stopes by this amount. On the other hand, heavy ground will increase the theoretical amount of timber required.

These conditions are indicated by the results at the Copper Queen mine, where, in 1907, 18 tons of ore was produced per set, showing a considerable overbreakage, and 300 bd.ft. of timber was used per set, which indicates the use of considerably more timber than required in a 12x12-in. set. In those days most stopes required the use of the larger-sized timber, and much reinforcing timber was used, which is now contrary to modern square-set practice. Table III shows the number of board feet and powder per ton and the tons per man-shift for various mines using the square-set method of mining.

Detailed Cost of Square-set Stoping—The labor costs of square-set stoping can be divided into four parts: (1) Drilling and blasting, (2) mucking, (3) timbering, (4) filling.

The following labor analysis is made upon the basis that the square-set contains 200 cu.ft. and that the ore has a specific gravity of 12.5 cu.ft. per ton, which gives 16 tons per square-set. The assumptions are made that ground conditions are good and that the ore requires no sorting. Wages are taken at $5.25 for drillers and $4.75 for muckers and a miner and mucker are assumed to work together. In average ground 35 to 50 ft. of hole should be sufficient to remove the ground for one

Table IV—Approximate Labor Cost, Square-Set Stoping

	Shifts	Rate	Cost Per Ton
Drilling and blasting	$0.6–0.8	$5.25	$0.20–0.26
Timbering	0.3–0.4	5.25	0.19–0.25
Timbering	0.3–0.4	4.75	
Mucking	0.80–1.0	4.75	0.24–0.30
Total			$0.63–0.81

Table III—Square-Set Data

Mine	Refer.	Per Ton Board Feet Stope	Board Feet Total	Pounds Powder Stope	Pounds Powder Total	Tons Per Shift Stope	Tons Per Shift Underground	Tons Per Shift Total	Remarks
Silver King Coalition, Utah	A.I.M.E. Vol. 72	3.2	3.9	0.82	1.86	....	1.73	0.90	Apr.-Nov., 1922
Silver King Coalition Utah	I.C. 6371	(25c.)	(34c.)	....	3.61	2.04	1.37	0.99	1929
Tintic Standard	I.C. 6360	19.7	25.5	0.42	1.06	....	1.11	0.95	Nov., 1929
Park-Utah, Utah	I.C. 6290	.....	16.5	....	1.53	4.14	2.26	2.00	1928
Bunker Hill & Sullivan	A.I.M.E. Vol. 72	.....	15.2	....	0.57	4.23	3.55	2.86	1921
Hecla & Star Mullan, Idaho	A.I.M.E. Vol. 72	.....	12.3 1.3 lin.	0.32	....	6.00	3.50	2.40	1922
Hecla & Star Mullan, Idaho	I.C. 6232	.....	11.2 1.2 lin.	0.53	....	6.10	4.00	....	1929
Morning, Mullan, Idaho	A.I.M.E. Vol. 72	12.3 1.2 lin.	.....	....	1.69	....	....	1.97	1922
Morning, Mullan, Idaho	I.C. 6238	12.75	.....	0.69	0.96	2.77	1.91	....	1928
Page, Page, Idaho	I.C. 6372	.....	13.1	....	1.40	....	2.90	....	1928
Ground Hog, N. M.	I.C. 6377	5.2	8.1	0.63	0.90	3.74	1.68	1.19	Jan.-Mar., 1930
Butte, Mont.	A.I.M.E. Vol. 72	.....	22.0	....	1.07	5.05	2.01	1.82	80% square set, 1921
Copper Queen, Ariz.	Personal	16.6	.....	....	....	2.30	....	....	1907
Copper Queen, Ariz.	Personal	14.0	.....	0.54	....	6.50	....	....	1931
Calumet & Arizona, Ariz.	A.I.M.E. Vol. 55 1916	14.3 10.8 11.5 8.5	....	....	....	....	....	....	Oxide Sulphide Oxide Sulphide } Mitchell Method
United Verde Extension	I.C 6250	.....	18.5	0.39	....	4.84	2.47	2.00	1929
United Verde, Jerome, Ariz.	A.I.M.E. Vol. 72	.....	14.5	....	1.83	....	3.68	....	1922
United Verde, Jerome, Ariz.	M.C.J. Apr., 1930	12.52 11.0	....	0.60 0.60	....	4.70 5.00	....	....	Horizontal Rill
Old Dominion Globe, Ariz.	M.C.J. Dec. 1928	.....	14.0	0.80	....	4.70	....	....	1928
Old Dominion Globe, Ariz.	Personal	8.1	.....	0.38	....	3.74	....	....	1930
Argonaut, Mother Lode	I.C. 6311	.....	7.0 1.08 lin.	....	1.01	....	1.92	....	Sept., 1929
Plymouth Consolidated Mother Lode	A.I.M.E. Vol. 72	.....	20.8	....	0.93	3.93	2.33	1.70	1922
Pilares, Mexico	M.C.J. May, 1928	.....	17.4	0.29	....	3.48	....	....	1928
Bawdwin, Burma, India	A.I.M.E. Vol. 72	.....	17.2	....	0.36	1.23	0.61	0.41	1921

square-set. Table IV shows approximate costs under the foregoing conditions.

At the Park-Utah mine (I.C. 6290), Park City, Utah, "square-set stopes are contracted for at $9 to $12 per square-set. Contractors are required to drill, blast, stand and block timber, place mining floor, rig up scraper and scrape ore into chutes." The cost per ton varies from $0.61 to $0.81. Filling is done on company account.

At the Calumet & Arizona mine (A.I.M.E., Vol. 55, p. 134), Bisbee, Ariz., in 1916, the labor cost of drilling, blasting, mucking, and timbering in

Table V—Labor Costs, Square Set Stoping, United Verde Mine

Labor	Cost Per Ton Horizontal	Rill
Miners	$0.34	$0.46
Plugger and carmen	0.66	0.54
Wastemen	0.12	0.11
Timbermen	0.27	0.18
Miscellaneous	0.05	0.04
Totals	$1.44	$1.33
Average tons per man in stopes	4.7	5.0
Powder per ton	0.6	0.6
Board feet timber per ton	12.5	11.0

square-set stopes with average oxide and sulphide ore was $0.60 and $0.49 per ton. If these costs be adjusted to the 1930 wage scale, the labor costs would be $0.79 per ton for exide ore and $0.64 per ton for sulphide ore.

If any substantial amount of sorting is done, this will increase costs appreciably. The cost is raised by the reduced tonnage of ore produced from the set, and also by the additional cost for sorting. Hard boulders in the ore will also add to the cost per ton. The figures presented in the preceding paragraph are representative only when conditions are ideal.

At the United Verde (*Min. Cong. Jour.*, April, 1930), Jerome, Ariz., where the ore is hard and boulders have to be drilled and blasted, costs are very much higher. Costs for both the horizontal and rill method are given in Table V.

Tables VI, VII, VIII, and IX show the labor and supply costs of square-setting at the Magma mine, Superior,

Table VI—Direct Stoping Costs

Labor	Magma(a) (Ariz.)	Pecos(b) (N. M.)	Cananea(c) (Mexico)
Mining	$0.460	$0.620	$0.571
Mucking	0.050	0.570	
Filling	.....	0.180	
Bonus	0.080		
Timbering	0.530	0.790	
Totals	$1.120	$2.160	$0.571
Supplies			
Explosives	$0.060	$0.150	$0.073
Timber	0.480	0.400	0.629
Miscellaneous	.....	0.120	
Waste fill	0.130	.....	.....
Totals	$0.670	$0.670	$0.702
Grand totals	$1.790	$2.830	$1.273
Filling	.....	.....	0.170
Air, drills, incidentals	0.141	0.085	0.185
Totals	$1.931	$2.915	$1.628

(a) I. C. 6138 (b) I. C. 6368 (c) I. C. 6247

Ariz., the Pecos mine of the American Metal Company of New Mexico, and of the Colorada mine, at Cananea, Sonora, Mexico.

Table VII—Direct Costs of Square-Set Stoping, Old Dominion Mine

Labor	Cost Per Ton
Drilling and blasting	$0.34
Mucking	0.17
Timbering	0.41
Filling	0.41
Total labor	$1.33
Supplies, incidentals	
Timber	0.28
Explosives	0.08
Air, drills, steel, incidentals	0.12
Miscellaneous	0.05
Supervision	0.06
Stope development	0.29
Total	$2.21
Board feet timber per ton	8.06
Pounds powder per ton	0.38
Tons per man-shift in stopes	3.74

My notes on the cost of square-set stoping at the Copper Queen, Bisbee, Ariz., in 1907, are given in Table VIII. The present costs of square-set stoping at the Copper Queen, as ascertained by correspondence, are given in Table IX. The timber required per set and the labor cost per ton for this mine, as shown in these two tables, furnish striking proof of the evolution through which square-setting has passed. The old idea that square-sets were used to

Table VIII—Cost, Square-Set Stoping, Copper Queen, 1907

	Cost Per Ton
Labor on ore	$0.984
Labor on dead work	0.562
Total labor	$1.546
Timber and lagging	0.550
Total	$2.096

Table IX—Present Cost, Square-Set Stoping, Copper Queen Mine

Labor	Cost Per Ton
Drilling and blasting	0.37
Mucking	0.58
Timbering, filling	0.19
Timber	0.45
Explosives	0.14
Air, drills, steel, incidentals	
Miscellaneous	0.32
Supervision	0.12
Stope development	0.30
Total	$2.47
Total labor, 1907	$1.546
Total labor, 1931	$1.140
1931	
Board feet per ton	$14.00
Pounds powder per ton	0.54
Tons per man-shift in stopes	6.5

support ground has given place to one which holds that the real function of the square-set is to serve as a scaffolding to permit the mining of ore and that it affords only a minimum amount of support to ground overhead. Recognition of this fact has allowed a great reduction in the amount of timber used. The labor costs per ton have thereby been greatly reduced and other innovations have lowered them still further, so that even though the wage scale has increased 50 per cent, the labor cost per ton today is less than it was 24 years ago.

My own experience indicates that the lowest costs of square-setting can be obtained by working long narrow stopes, with 10x10-in. timber on the sill floor and possibly the second floor, but with 8x8-in. timber on all other floors. With the lighter timber in heavy ground rapid working and close filling become necessary. Only under exceptional conditions can square-set timber be recovered economically from a stope.

Waste Filling Costs—In most mine-accounting systems the cost of waste filling is seldom recorded as a separate item. It is usually one of the largest items in square-set mining, although seldom recognized as such. In most discussions of costs of stoping by methods that call for filling, the statement is simply made that filling comes from development. Not often is a stope

Table X—Cost of Waste Filling, United Verde Mine, 1918

	Cost Per Ton
Mining per ton of ore	$0.049
Delivering to stope per ton of ore	0.254
Distributing in stope per ton of ore	0.246
Total	$0.549

so located that it is convenient to the development work. Perhaps waste is hoisted from a lower level and trammed up-grade from the shaft. If it comes from an upper level, transfer must be made through a raise. If waste is not being delivered fast enough, it is obtained by driving raises or drifts into the walls of the stope. In some operations such work has a real prospect value, but in others this is not so. A tendency may be evidenced to push development work convenient to stopes needing waste rather than more meritorious work less conveniently located. Under certain conditions mining of waste on surface may be necessary to provide sufficient filling. This method of obtaining waste has been applied at the United Verde (A.I.M.E., Vol. 66, p. 165), where it was done with the costs given in Table X. These costs were for 1918. The cost of mining and delivering waste to the stopes in 1920 was $0.207 per ton of ore. With a cost of $0.247 per ton of ore for distributing in stopes, the total filling cost is $0.454 per ton or ore produced. In 1926 (*Min. Cong. Journ.*, Sept., 1927), the cost of spreading waste in square-set stopes was reduced to $0.09 per ton of ore.

At Matahambre (*E.&M.J.*, March 2, 1929) a very economical method of delivering filling to the stopes and spreading has been devised. The sand from the mill tailing is separated from the slimes and delivered to the stopes through a 2-in. pipe line. This was

Table XI—Costs per Ton, Square-Set Mining

	Park Utah, I.C. 6290	Tintic Standard, I.C. 6360	United Verde Extension, I.C. 6250	United Verde, A.I.M.E., Vol. 66	Argonaut, I.C. 6311	Morning, I.C. 6238	Hecla & Star, I.C. 6232	Bunker Hill & Sullivan, A.I.M.E., Vol. 72	Bawdwin, Burma, A.I.M.E., Vol. 69
Labor	$1.437	$3.050	.....	$1.976	$1.746	$1.550	$1.030	$1.225	$0.610
Supervision	0.060	0.323	.....	0.243	0.052	0.142	0.080	.....	0.180
Explosives	0.112	0.114	.....	0.096	0.148	0.114	0.112	.....	0.210
Timber	0.423	0.841	.....	0.612	0.335	0.322	0.357	.....	0.280
Compressed air, drill repair, steel	0.101	0.574	0.244	0.344	0.013	0.091	0.070	.....	.....
Power	0.091	.....	.....	.....	0.088	0.098	.....	.....	.....
Miscellaneous	0.202	0.071	.....	0.092	0.057	0.054	0.101	0.554	0.380
Total stoping	$2.426	$4.973	$1.954	$3.363	$2.439	$2.371	$1.750	$1.779	$1.660
Development	0.828	1.143	0.611	0.616	0.165	0.442	0.638	.....	0.520
Transportation	0.282	1.056	0.508	0.565	0.519	0.530	0.322	.....	0.680
General	0.583	0.424	0.474	0.370	0.577	0.698	0.227	.....	0.370
Miscellaneous	0.543	1.888	0.743	1.403	0.291	0.204	0.360	1.022	0.040
Total	$4.662	$9.484	$4.290	$6.317	$3.991	$4.245	$3.297	$2.801	$3.270
Year	1928	Nov. 1929	1928	1918	1929	1928	1928	1922	1921

done in 1928 at the following cost: Labor, 15.6 cents per ton of fill; power, 1.1; supplies, 18.6; total, 35.3c. By substituting a 3-in. rubber-lined pipe, according to a preliminary estimate, costs could be reduced to 20c. per ton of material. (This has since been accomplished—Ed.) Waste filling obtained from surface had formerly cost 55c. per ton of material. The cost of filling at this mine per ton of ore varies from 15 to 20c.

The Magma mine (I.C. 6168) reports a supply cost chargeable against waste fill of 13c. per ton of ore. The Pecos mine (I.C. 6368) reports a labor cost of 18c. chargeable against filling. At Cananea (I.C. 6247), the filling charge at the Colorada mine is 17c. per ton of ore. At the Tintic Standard (I.C. 6360), filling costs 33c. per ton of ore. From data already mentioned filling at the United Verde cost about 30c. per ton of ore. Filling costs at Bawdwin mine (*Trans.* A.I.M.E. Vol. 69) Burma, are 38c. per ton.

At mines where filling costs are kept only direct costs are usually recorded. Stope waste raises are charged directly to the stope, but when a waste storage or transfer raise is driven, to supply a whole level or a number of stopes, it is usually charged to development. This cost is really a prepaid development charge which is a part of the filling cost and should be distributed over the waste handled. However, such a distribution is not feasible, as most cost sheets are already sufficiently complicated. Nevertheless, in the selection of a mining method, the analysis of the expected costs should include the direct cost of filling and also the indirect costs. In addition to the raises mentioned, these include maintenance of drifts, track, and raises. Often they are rather high, especially where ore is being mined in soft, heavy ground. In any method requiring the filling of stopes I believe that 25c. to 50c. per ton should be allowed for filling charges, dependent upon the character of the ground being mined, the method of obtaining fill, the means of delivery to the stope, and the method of spreading in stopes. Whenever a stope is being brought up under a stoped-out area on a level above, a considerable amount of back-filling is necessary, which in nearly all such mining has to be done by hand shoveling. This is a very expensive method of filling, which, however, applies to only a part of the stope.

In Tables XI and XII are presented the costs per ton of square-set mining and stoping in numerous mines where the square-set method of mining is the most important one in use. Most of these mines use the typical or conventional square-set. At the Morning mine and at the Hecla and Star mines a timber-drift set is used in place of the regulation square-set. All except one of the mines are in the United States, the Bawdwin mine, in Burma, being added for comparative purposes. The wages at the Bawdwin are about 48c. per shift. Although the cost per ton there is lower than that at most of the other mines, it is not nearly proportional to wages.

Estimated Costs of Square-Setting—An attempt is made here to build up a cost of square-set mining based on operations in various parts of the United States. Each item of cost is given a rather wide range to cover extreme cases. The minimum and maximum total costs are not supposed to be representative costs for any one mine, because no one mine has advantages which gives it all of the minimum individual costs; nor is it usual for another mine to be confronted with high costs in each operation. As an example, if drilling, blasting, and explosive costs are low, owing to soft ground, timbering and filling costs are likely to be high. The conditions may be reversed in hard ground.

In conclusion, I would point out that although great differences occur between the total costs, many individual items show a close similarity. The Tintic Standard and Silver King Coalition are striking exceptions, and these costs are not referred to in the following remarks. The labor costs of stoping do not differ greatly. That of the United Verde is the highest, which is attributed to the fact that the ground is generally hard and requires the breaking and blasting of boulders. The labor cost at the Hecla is the lowest, which is accounted for by the fact that part of the tonnage comes from shrinkage stopes, a circumstance not typical of square-setting. Explosives costs in many operations are very similar and average about 12c. per ton. The timber cost is about 40c. per ton. Quite a difference is to be noted between

Table XIII—Estimated Direct Stoping Costs per Ton

	Minimum	Maximum
Drilling and blasting	$0.20	$0.40
Mucking	0.20	0.35
Timbering	0.20	0.50
Filling	0.25	0.50
Explosives 0.5 to 1 lb.	0.08	0.15
Timber 9 to 22 bd.ft.	0.35	0.85
Air, drills, steel, repairs	0.10	0.30
Miscellaneous	0.07	0.20
Total	$1.45	$3.25
Supervision	$0.05	$0.15
Stope development	0.25	0.50
Total	$1.75	$3.90

Table XII—Cost of Square-Set Stoping

Stoping cost	Ground Hog, Asarco Mining, Vanadium, N. M., I.C. 6377	Silver King Coalition, Utah. I.C. 6371	Page, Federal Mining & Smelting, Idaho, I.C. 6372	Bunker Hill & Sullivan, I.C. 6407	Old Dominion, Globe, Ariz. Personal	Copper Queen, Bisbee, Ariz. Personal	Magma, Superior, Ariz. I.C. 6138	Pecos, N. M. I.C. 6368	Cananea, Mexico, I.C. 6247
Labor	$1.197	$2.454	$1.430	$1.419	$1.33	$1.14	$1.12	$2.16	$0.57
Supervision	0.065	0.130	0.150	0.057	0.06	0.12	....	....	....
Explosives	0.058	0.173	0.110	0.167	0.08	0.14	0.06	0.15	0.07
Timber	0.120	0.249	0.260	0.321	0.28	0.45	0.48	0.40	0.63
Compressed air, drill repairs, incidentals	0.053	0.472	0.110	0.209	0.12	0.32	0.14	0.09	0.19
Power	0.034	.....	0.050	.....	....	....	....	....	....
Miscellaneous	0.235	0.024	0.040	0.241	0.34a	0.30a	0.13	0.12	0.17
Total stoping cost	$1.762	$3.502	$2.150	$2.414	$2.21	$2.47	$1.93	$2.92	$1.63
Development	1.393	2.086	1.170	0.251	....	....	....	....	....
Transportation	0.628	1.848	0.160	.....					
General	0.080	.....	0.180	.....	....	....	....	....	....
Miscellaneous	0.127	0.309	0.160	1.232	....	....	....	....	....
Total	$3.990	$7.745	$3.820	$3.897	.....	.....	.....	.....	.....
Year	1930	1929	1928	1928	1930	1931	1929	1930	1929

a Includes stope development

the costs of compressed air, drill repairs, and steel. This is probably due to the methods of keeping and segregating these costs. With few exceptions the total cost of stoping varies little for the different mines. It ranges from $1.75 to $2.43. The United Verde cost is high, owing to the method of segregating supervising costs, and to high timber, compressed air, and drill and miscellaneous costs. Development costs differ but little, with the exception of those at the Argonaut and Morning mines. The difference in the latter instances is natural because development costs less in working lodes than in mining the irregular replacement deposits of the other mines.

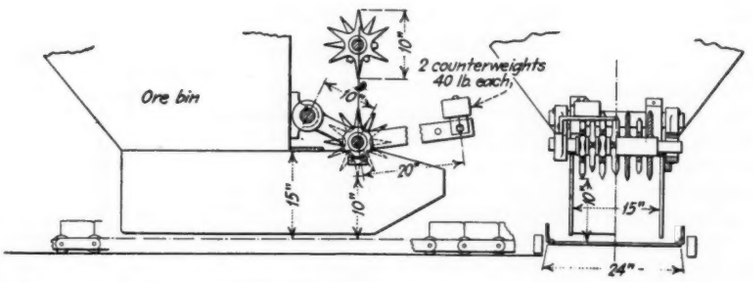
With the exception of the Park Utah and the Hecla operations, underground transportation costs differ little. The greatest variation in any cost is the item "miscellaneous and surface." To a certain extent this cost is a catch-all for undistributed costs and often includes items not directly chargeable to the mining operation. In the tables, I have combined "miscellaneous and surface" for all mines as "miscellaneous."

Square-set mining is the most expensive of all methods. This is not the result entirely of the method itself, as low costs can be obtained, as at Bunker Hill & Sullivan. Its cost compares favorably with those of cut-and-fill methods. The costs of actual breaking and delivering ore to the stope chutes are not comparatively high. That square-set mining shows high costs is due, principally, to the conditions under which it is employed. In many operations the orebodies mined by it are irregular in size and shape and the ore often requires much sorting. In some cases the ground is exceedingly heavy. Where the surrounding ground is also heavy, maintenance of drifts and raises is costly. As an auxiliary to the cut-and-fill method in mining the ore under a worked-out stope on a level above, the method is at some disadvantage from the viewpoint of cost.

The U. S. Bureau of Mines estimates that nearly 8,000,000 tons in the United States is mined annually by the square-set method. The larger part of this tonnage is mined by this method because it permits selective mining in ground which requires support. Its only competitor is top-slicing.

■ ■ ■

Mechanical Gate for Ore Bins



CLOGGED ORE PASSAGES, damaged conveyor plates, and interruption of crushing operations often resulted from lack of proper control gates on the ore feed bins, writes R. Limonet, smelter engineer, Compagnie Du Boleo, Santa Rosalia, Baja California, Mexico. Coarse ore from the mines, in sizes from 1 to 14 in., is fed through 10x15-in. openings in the feed bins over a Jeffrey metallic conveyor to a Pennsylvania hammer crusher. No difficulties were experienced with ore particles not exceeding 14 in., but larger pieces would invariably choke the bin gate; if forced through, they would damage the plates of the conveyor. To overcome this difficulty each bin gate was equipped with a movable gate, as shown in the accompanying sketch. The device consists of a steel axle, with seven revolving star-shaped knives of ½-in. boiler plate; two mild-steel levers, keyed to a short shaft resting in two end bearings bolted to the ore bin wall; and two 40-lb. counterweights. The scheme of operation is as follows: The pressure exerted by large ore particles while passing through the bin opening automatically raises the lever-like gate, which remains in this position until the oversize mass of ore has been forced upon the conveyor, after which the two counterweights return the gate to its normal position, to prevent a sudden flow of fine material. The rotating knives aid in maintaining a uniform feed of ore to the crusher. Conveyor efficiency may be increased materially by angle irons fastened at intervals to the conveyor plates.

■ ■ ■

Shaft-Sinking Equipment—I

Lucien Eaton

THE equipment used for shaft-sinking is of a more or less temporary nature; but, by proper planning, much of it can be used later as permanent equipment. If the shaft is a deep one, this is not important, because shaft-sinking is rough work, and much of the equipment will be nearly worn out when the shaft is finished. The larger and deeper the shaft, the more elaborate the equipment.

Much of the equipment needed is standard, except for the number of pieces required, whether the shaft be inclined or vertical, large or small, but great depth always requires relatively large hoisting equipment. However, it is usually feasible to sink the upper portion of the shaft with light equipment, and then to change over to the permanent hoists erected meanwhile.

Avoidance of delay is important if the shaft is to be put down quickly, and a quick start is almost as important as later speed. The first 20 to 30 ft. are likely to be sunk by hand anyway, and this can be done while the plant is being assembled. Even during the construction and erection of the sinking equipment, it may be possible to sink shaft at night, while the construction crew is not working, without interference with either operation. This procedure also makes possible the gradual assembly and organization of a shaft crew, preventing delay after the equipment is erected.

The equipment needed for shaft-sinking may be listed under the following heads: (1) Power plant; (2) air compressor; (3) headframe; (4) hoisting engine; (5) hoisting and loading equipment; (6) disposal of waste; (7) timbering, or galloway, stage; (8) drills and drill steel; (9) pumps; (10) ventilation; (11) lighting; (12) change house or dry; (13) sanitation; (14) shops; (15) office and warehouse; (16) transport; (17) explosives; (18) housing.

If the shaft is sunk in the footwall, the permanent equipment will in all probability be on the footwall side, and the temporary equipment should therefore be set up on the hanging-wall side, or in such a position that it will interfere as little as possible with the erection of the permanent equipment. The boiler, hoist, and air compressor, especially in a small plant, should be erected close together, preferably under one roof, to minimize the cost of inclosure and heating and the labor of attendance. In a small steam plant, the hoisting engineer may fire the boiler and run the air compressor, as well as operate the hoist. Concentration of all departments into the fewest buildings is economical and convenient.

If electric power is obtainable within a few miles, good policy dictates the advisability of carrying a pole line to the shaft site as soon as possible, instead of setting up a boiler plant. In such case a transformer station and switchboard must be provided. If electric power is not available, a steam plant must be built, unless the shaft is small or shallow, in which circumstance a gasoline-driven hoist and air compressor will probably be more economical. Adequate supply of fuel at reasonable cost and water suitable for use in boilers are, of course, essential. The absence of either may be the deciding factor in selecting a power plant.

The locomotive boiler with an individual steel stack is the most convenient type, and is reasonably efficient. Adequate capacity should be provided. Two smaller boilers are preferable to one large boiler, for convenience in in-

This is the third of a series of articles on mine equipment.

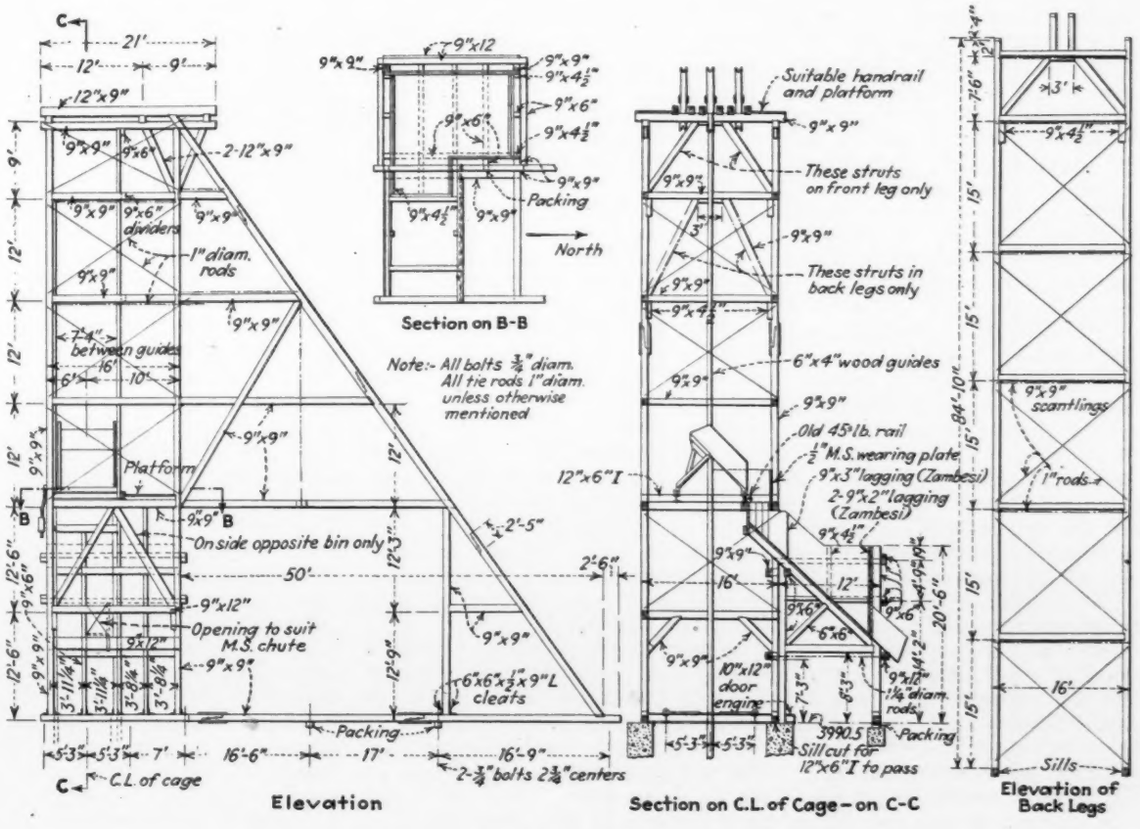
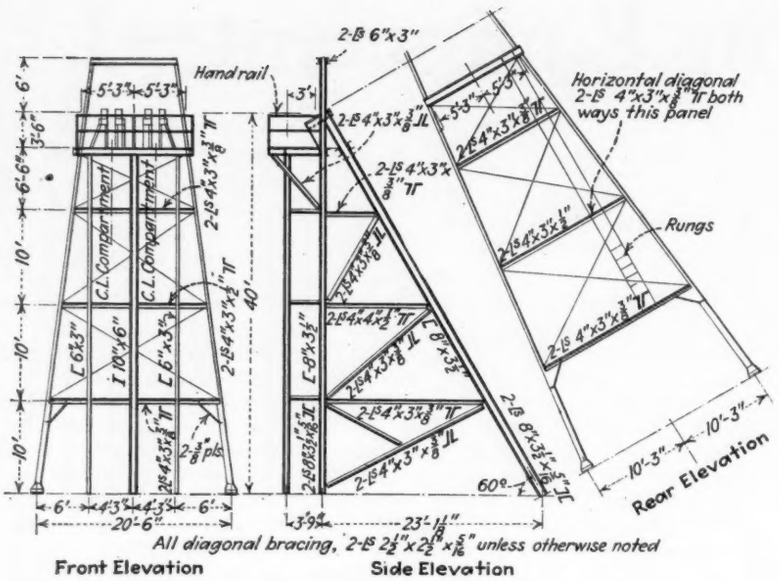


Fig. 12a—Timber headframe for sinking, Roan Antelope Copper Mines



stallation and operation. Later, one of these boilers can be used as the heating plant for the dry or change house, and the other one possibly as an air receiver. Two 75-hp. boilers make a convenient unit. If this has insufficient capacity, the size of the boilers can be increased to 100 or 125 hp. A supply of good water is essential, and a water tank, holding several hours' supply, should be provided. Injectors as well as a feed pump are desirable, and a feed-water heater will prove an economy. Provision for storage of fuel need not be elaborate, but a sufficient supply should be on hand to avoid delays.

The air compressor will be the greatest power consumer, unless the shaft is very wet, and its size will depend on the size of the shaft and the number of drills and pumps to be operated. A shaft 6x12 ft. outside of timbers will probably use four drills, and a 500-cu.ft. air compressor should be large enough. For a shaft 12x16 ft. outside of timbers a capacity of 1,000 cu.ft. per minute will be sufficient, if little pumping be done with air. For a 20-ft. circular shaft I would recommend 1,500 cu.ft. of capacity, although to operate with less may be possible. If the air compressors are electrically driven, good practice dictates the use of duplicate units—one, two, or three 500-cu.ft. machines in the operations just mentioned, unless the capacity wanted is the same as one of the units of the permanent compressor plant. In that case, one of these units should be used. The smaller steam-driven air compressors are not as efficient as the larger units; and so, if steam is the power used, the advantages to be obtained by duplication of small units is neutralized. Single-stage compression should be used only in machines of less than 300 cu.ft. per minute capacity.

An air receiver should be provided, with a capacity one-seventh the volume of free air compressed per minute. For example, a 500-cu.ft. compressor requires a 72-cu.ft. receiver, which should be fitted with a blow-off valve and a safety valve. Almost any small boiler will serve as a receiver, or, if large pipe is available, such as 12-in. pipe, for the counterweight, a few lengths, closed at the ends with blank flanges made of plate, may be used.

The diameter of the main air pipe will be 3 in. for the 500-cu.ft. unit, 4 in. for the 1,000-cu.ft. unit, and 5 in. for the 1,500-cu.ft. unit. This will be standard pipe with standard malleable flanges or Victaulic couplings.

Ordinarily, the headframe used for sinking is a simple affair, of wood; but, if the shaft is large and deep, a rather elaborate structure may be justified. At some large shafts the headframes have been as high as 100 ft., with sheaves for four or more skips. When practicable, time is saved if the permanent headframe can be erected while sinking is proceeding. The sinking headframe may be designed in such a way that it will not conflict with the principal members of the larger structure, which can be built over it and around it without interfering with sinking operations. This is particularly important if the main hoists are to be used during the latter part of the sinking period.

If the collar of the shaft is to be raised above the ground level, the sinking headframe should be supported on beams above this elevation, so as not to interfere with the construction of the collar. Under almost any circumstance its supports should be independent of the shaft and far enough away from it to be unaffected by sinking operations.

The sinking headframe may be little more than a tripod or it may be a fairly elaborate structure. In some districts the A type of frame is popular, but a four-post frame with two back legs is as simple to erect as any, and can readily be made independent of the permanent headframe which is to be erected over it. Examples of sinking headframes are shown in Fig. 12.

Light headframes can be made with members built up of 2-in. plank, as is the custom in the Tri-State zinc district; but commoner practice is to use timbers from 8 to 10 in. square. Tierods will reduce the timber required.

Height of headframe will depend on: (1) The elevation at which waste rock can be disposed of; (2) the height required for the rock car and the pocket or chute above it; (3) the height required for the skip, car, or bucket to dump; (4) the height of the crossheads, cage, or skip bail and extension guides; (5) the clearance allowed under the sheave.

Fig. 12b, above—Portable steel headframe, 40 ft. high, Roan Antelope Copper Mines

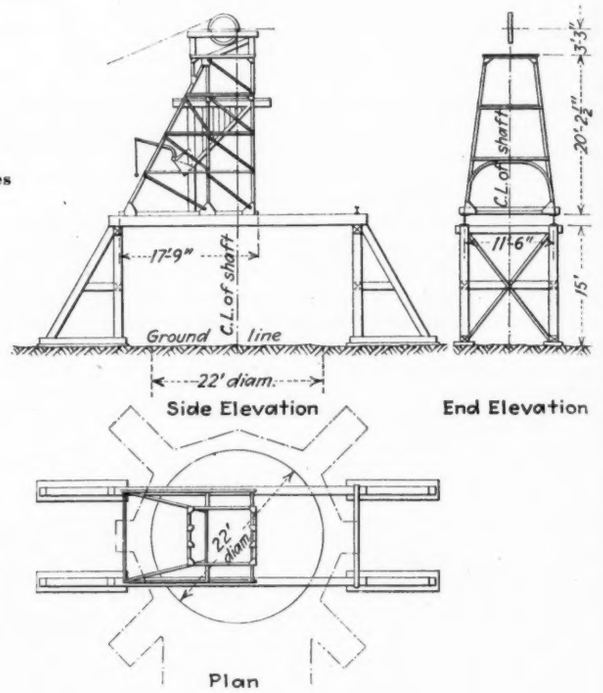


Fig. 12c — Temporary sinking headframe, Witwatersrand, South Africa

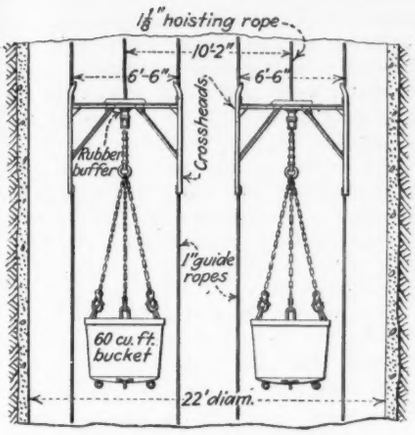


Fig. 14 — Crossheads and sinking buckets on rope guides in 22-ft. diameter circular shaft

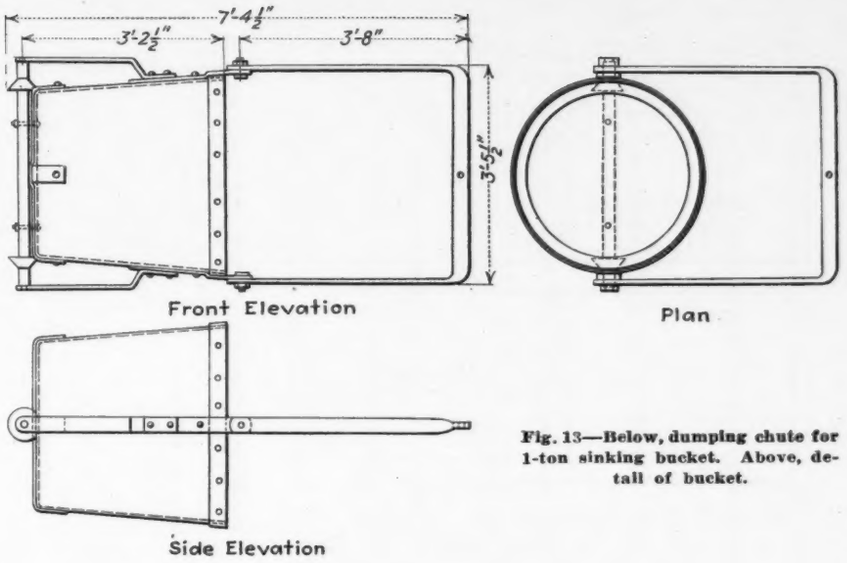
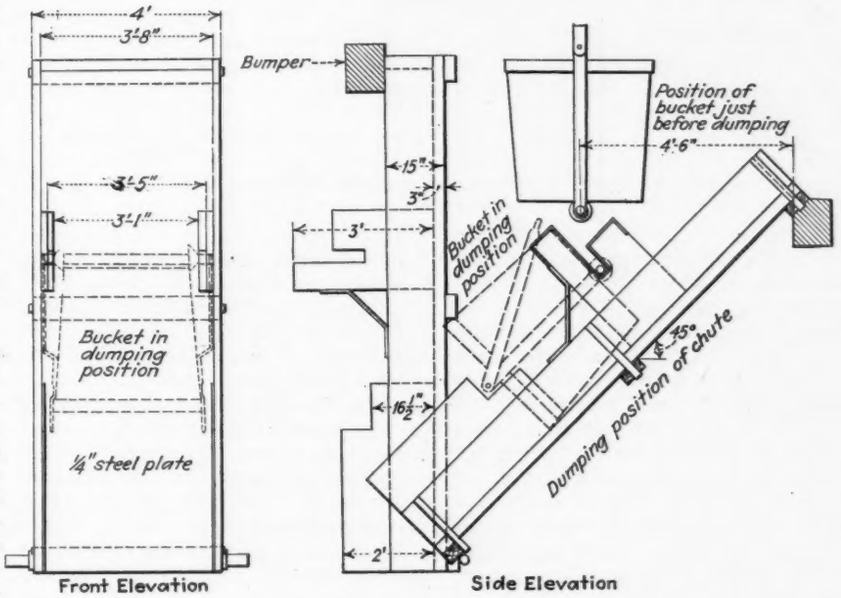


Fig. 13—Below, dumping chute for 1-ton sinking bucket. Above, detail of bucket.

If the topography is such that waste rock can be dumped at the elevation of the shaft collar, the waste is usually trammed by hand. If the topography is level, the cars may have to be trammed on a trestle, at a higher elevation, or hauled up an incline.

The height of the rock car will be from 3 to 5 ft., depending on its size, and the lip of the chute should be 1 ft. higher. If the car is of the same capacity as or larger than the skip or bucket, no storage need be provided, and the skip or bucket can be dumped into the car, directly or through a chute. Otherwise, a pocket should be provided holding two or more skip loads. A chute may require only 3 ft. of headroom, but a pocket will be 5 to 10 ft. or more high. The height required to dump the skip or bucket depends on design and size, and varies from 5 to 15 ft. If sinking is done with a cage, the height required for the chute or pocket and for the dump may be omitted.



Nearly everywhere, when the shaft has reached 300 ft., a crosshead is required for the bucket, and good practice suggests its use at much shallower depths. The headroom needed for a single crosshead is 2 to 5 ft. Skips and cages must have extension guides or double crossheads, and the headroom required for them may vary from 15 to 60 ft. Clearance under the sheave need not be large for sinking operations, and the minimum may be assumed as the diameter of the sheave.

The collar of the shaft should be completely covered except the hoisting compartments, and these should be protected by a tight fence with suitable gates, or by doors. These doors should be closed whenever the bucket is dumped. They are substantially built, and are so designed that they slope outward, so that anything that falls on them will be diverted away from the shaft. At small shafts they may be operated by hand; but at large shafts, compressed-air cylinders or air engines should be used.

If the disposal of waste is at an elevation higher than the collar, and if a bucket is used, doors should be provided at this higher elevation. Bucket dump and door are frequently combined.

Head sheaves are usually of cast iron for diameters up to 4 ft., but larger sizes are more likely to be bicycle sheaves. The diameter, usually smaller with respect to the size of the hoisting rope than it is in the permanent plant, varies from 48 to 60 times the diameter of the rope.

At a small shaft, or in the first 100

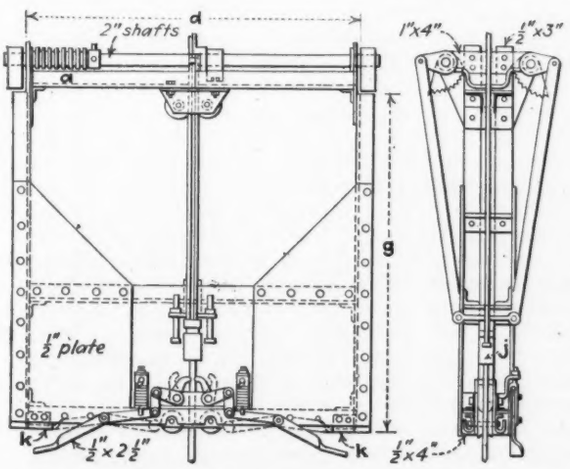


Fig. 15 — Safety crosshead for sinking bucket. (U. S. Bureau of Mines, T.P. 276.)

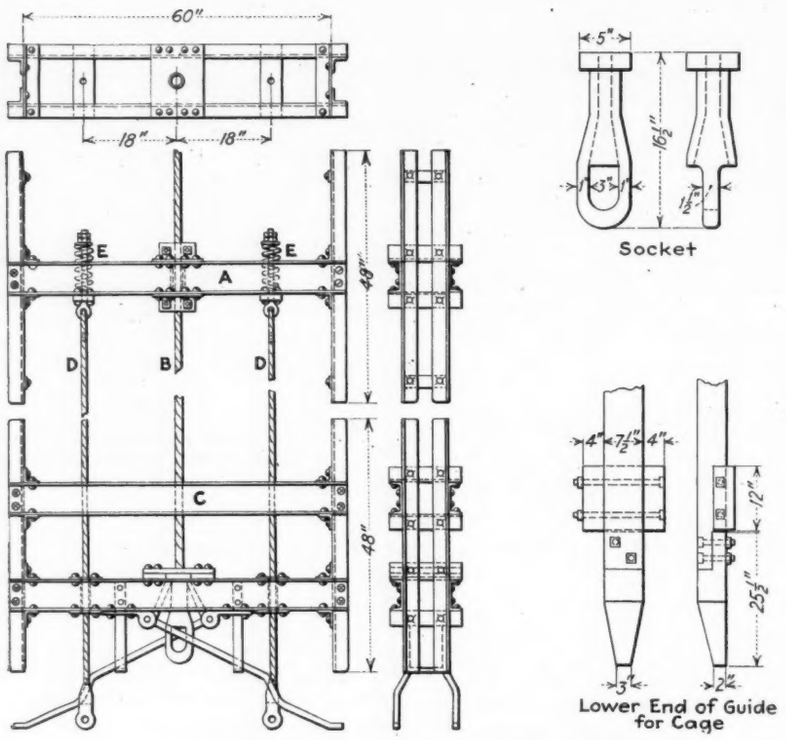


Fig. 16—Double crosshead for use with bucket or sinking cage

ft. of a large one, hoisting can be done satisfactorily with a single-drum, geared hoist, having cylinders 6½ in. x 8 in. or 7 in. x 9 in., driven by steam or compressed air. The drum, with deep flanges, will be 14 in. diameter, with friction clutch and foot brake, and should have a ratchet ring and ratchet bar at one end, so that the bucket can be held securely at any point without the use of the brake. Such a hoist will handle a ½-ton bucket at a speed of 300 ft. per minute. If the shaft is large, it can be used later for handling pumps, drill manifolds, timber, and pipes. If electric power is available, an electric hoist driven by a 25-hp. motor may be substituted. The regular sinking hoist should be larger, but the size will vary with size and depth of shaft.

Assume that the shaft is to be vertical and 1,000 ft. deep, and that in cross-section it is either rectangular, 12x16 ft. outside of timbers, or circular, 17 ft. internal diameter, which are equivalent sizes. In this type of shaft the hoisting equipment would be a 2-ton bucket or a cage with a 1½-ton car on it. The hoist would have a 5-ft. drum, geared to a 200-hp. a.c. motor or a 150-hp. d.c. motor, and the rope speed would be 600 ft. per minute.

If the section of the shaft were long and narrow, two 1-ton buckets hoisted by two single-drum hoists with drums 4 ft. in diameter geared to a 125-hp. a.c. or a 100-hp. d.c. motor with a rope speed of 500 ft. per minute would be suitable. If steam is used the engine cylinders would be 12 in. and 10 in. in diameter, respectively.

For sinking the large shafts on the Witwatersrand, the standard sinking hoist has two drums, 8 ft. diameter and about 2 ft. across the face, geared about 3 to 1 to two 16½x33-in. steam engines. This hoist will handle a 2-ton skip or a 3-ton bucket at a speed of 1,500 ft. per minute. A prejudice is in evidence in this district against electric hoists for sinking, based apparently on the customs of blasting with fuse instead of electricity, and of not providing ladders in the shafts.

The standard practice has been to use one 3-ton bucket in circular shafts, and 2-ton skips with extension guides in the rectangular shafts. Recent practice in circular shafts over 2,000 ft. deep has been to use two 3-ton buckets in balance.

The simplest form of signal in shaft sinking is usually the best, and this is ordinarily the pull bell or knocker. When the pull bell is used it should be an 8-in. or 10-in. gong with a spring release. The knocker is in the shape of a hammer that strikes loose plates both above and below. The bell cord is ⅞ in., 6x7 galvanized strand, and is wound on a reel near the collar of the shaft. It is connected by clips to a short piece of cord attached to the bell or knocker. By shifting this connection the bell cord is paid out as the shaft is deepened. Signals can be rung directly to the engine house, or they can be relayed to the hoisting engineer by the lander in the headframe. In small outfits the first plan is probably better, and the second at larger plants.

The bell cord should be so placed in the shaft that it can be reached at any time by men riding on the bucket or cage. It should be counterbalanced every 300 or 400 ft. Pull bells do not work so well at depths over 1,000 ft.; but, when this depth is reached, an intermediate level has usually been turned off, and the pull bell can be rung to this level, and relayed therefrom by electricity to the engine house. On account of the danger from stray currents when blasting by electricity, it is good policy not to attempt to signal from the bottom of the shaft by electric bell.

Buckets are used in shaft-sinking more generally than skips or cages, but both of the latter have their strong advocates, and they possess many advantages, especially in large installations. Four types of sinking buckets are used:

1. One in which the bail is attached at the rim: This type, common all over the world in sizes up to 1 ton, is dumped by means of a ring in the bottom to which a chain is hooked, or by some special device. This bucket has the advantage of stability.

2. One in which the bail is attached to the sides of the bucket below the center of gravity: The bucket is held upright by sliding rings on the bail, which engage pins on the rim of the bucket, or by folding lugs on the rim, which engage the bail. This design has the advantage of a low dumping height, but is not as stable as the first type, and requires more labor for dumping. It works well in sizes up to 1½ tons.

3. Special designs, having trunnions or other attachments near the bottom, which dump more or less automatically in special dumps: This type has the advantage of stability and rapid dumping, and requires little labor, even in the larger sizes with a capacity of 2 tons or more. A bucket of this type and its dump are shown in Fig. 13.

In operation, the bucket is hoisted above the dump, which is pulled into position by an air cylinder. When the bucket is lowered, the crosshead is supported on special chairs, and the trunnions engage slots in the sides of the dump. These slots throw the bucket off center and cause it to dump in the right direction. After dumping, the bucket is hoisted again, the dump is raised and held out of the way by a counterweight, the crosshead chairs are thrown out, and the bucket is lowered into the shaft. When a single crosshead is used, the bucket must be rotated by hand, until the trunnions are in line with the dump. If a double crosshead is used, this is unnecessary. This type of dump would work well with a South African bucket. The device is patented, but its use is offered freely to the profession.

In sinking inclined shafts, hoisting is often done with buckets that slide on skids made of round poles, laid on the footwall. Several ingenious devices have been invented for automatic dumping, most of which make use of trunnions on the bucket.

In another automatic type of dump is a large ring fastened to the bottom of

the bucket by brackets, and so arranged that, when the bucket is lowered upon the shaft door, this ring engages a hook on the door, and both bucket and door are overturned as the bucket is lowered. This dump is suitable for the smaller sizes of buckets, but the bottom ring increases the height.

Another somewhat similar device consists of a chain, 3 or 4 ft. long, with a ball or washer at the lower end, which is attached to a ring in the bottom of the bucket. A slot is provided in the middle of the shaft door, which, when the door is swung across the shaft under the waiting bucket, engages the chain, causing the bucket to dump when it is lowered. When the bucket is resting on the dirt at the bottom of the shaft, this chain usually tips it a little to one side, and makes loading difficult.

4. The South African type, in which a bridle, consisting of three chains and safety hooks, serves as a bail: At the bottom, in line with each lug to which a chain is attached, is a dumping ring, and the bucket is dumped by a hook in the same manner as the first type. This design has great stability, is easily dumped, and allows the bridle to be shifted quickly from one bucket to another at the bottom of the shaft. It is especially good for large buckets, as it permits of large capacity without too great height. The standard 3-ton bucket is 60 in. diameter and 44 in. high. This design is shown in Fig. 14.

Except in shallow shafts, less than 150 ft. deep, crossheads should be used to keep the buckets from striking the sides. The simplest type is a beam of wood with notches on the ends to fit the guides and a hole in the middle through which the rope passes. Such a crosshead is dangerous, as it is likely to hang up on the guides, and then fall, striking the bucket and injuring anyone who might be riding on it. Generally accepted practice indicates that the height of a crosshead should not be less than two-thirds of the distance between guides, and that the clearances of the guide shoes should not be less than $\frac{1}{2}$ in. nor more than 1 in. between the guides, and not less than $\frac{1}{4}$ in. nor more than $\frac{3}{4}$ in. across the face of the guides.

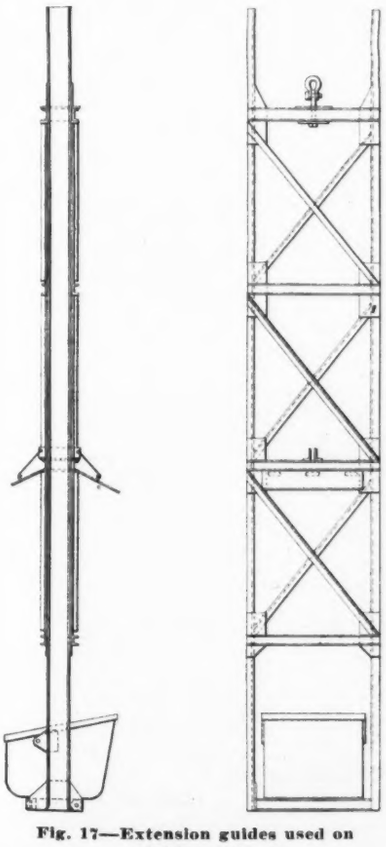


Fig. 17—Extension guides used on sinking skip

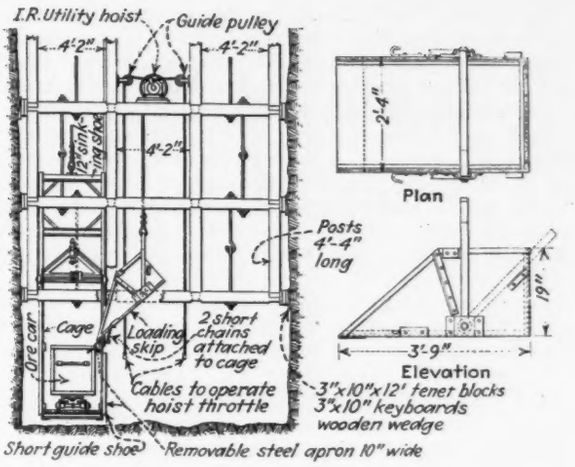


Fig. 18—Mechanical mucking with loading skip at Park Konold shaft, Park City, Utah

Crossheads with safety catches and bonnets have been designed, having catches arranged so that the bucket will not fall, even if the rope breaks. Such a crosshead, but without the bonnet, is shown in Fig. 15. The design shown is the Bryant crosshead, recommended by the U. S. Bureau of Mines. In steeply inclined shafts a trolley, which runs on an I-beam track, suspended from the hanging-wall plate, may be used if headroom is sufficient.

If the bucket must not rotate, and must always be in the same position for dumping, a double crosshead may be used, such as shown in Fig. 16. This device is not widely known, although it has been used on the Marquette Range in Michigan for sixteen or seventeen years. It was used in sinking the Athens and Holmes shafts in 1915 and 1916, and has been standard equipment since.

The double crosshead consists, as its name implies, of two crossheads, an upper crosshead (A), fast on the hoisting rope (B), and a lower crosshead (C), free to slide on the hoisting rope and on the two guide ropes (D), which extend from the upper crosshead to the bail of the bucket or cage. When the bucket is lowered into the shaft, the lower crosshead rests on top of the socket, to which the bucket is attached, and keeps the bucket from rotating or swinging. When the bottom end of the shaft guides is reached, the lower crosshead is stopped by and rests on blocks attached to the guides, and the catches that hold the bucket to the lower crosshead are released by these blocks. The bucket then continues to the bottom of the shaft, the only limitation being that the distance from the end of the guides to the bottom of the shaft must be less than that between the bucket and the upper crosshead. If this distance is long, considerable headroom is necessary in the headframe; and, for this reason, to use the double crosshead is not always feasible until the permanent headframe has been erected. The two guide ropes (D) need not be more than $\frac{3}{4}$ in. diameter, and they are kept in even tension by the springs (E) on the upper crosshead.

If the crossheads are of steel and are to be used in cold weather, the guide shoes should be lined with wood, so that they will not freeze to the guides, as might happen if they were left in the cold for any considerable time.

Skips save time in dumping, but more danger is involved of spilled dirt falling down the shaft than with buckets or cages. A typical design is shown in Fig. 17. Note that the back is made low, to facilitate shoveling, and that the lip is higher, to reduce spill in dumping. The length of the extension guides, which may be as much as 45 ft., depends on the distance the shaft is sunk below the ends of the shaft guides. Skips work equally well with double crossheads, by the use of which the distance below timbers can be increased and the weight of the skip reduced. A 2-ton skip with extension guides 32 ft. long weighs $2\frac{3}{4}$ tons, over half of which is in the extension guides.

In sinking an inclined shaft, hoisting is often done with skips with part of the back cut away to facilitate shoveling. The track is extended below the timber to the bottom of the shaft by means of sets of rails, which overlap and slide upon the rails of the permanent track.

In recent years several large companies have standardized cages for sinking, and their popularity is apparently increasing. When used with either extension guides or double crossheads, cages have many advantages. For both safety and convenience in handling men

and supplies they are superior to skips or buckets, and they require little headroom for unloading, with little danger of rock falling from them into the shaft.

A sinking cage should be of simple, open construction, designed so that the frame will interfere as little as possible with loading the car. It is usually provided with a bonnet and safety catches. The car, which the cage carries, should be low and broad, to facilitate shoveling. It may be of special design, or one of the standard underground cars may be used, if of the proper size. A typical sinking cage is shown in Fig. 18.

In operation, when the car is full, the cage is hoisted to the dumping point, the car is run off, dumped, and returned to the cage; or an empty car is put on in its place, as soon as it has left the cage. If the distance hoisted is short, the first plan is usually followed; if long, the second is preferable.

If the timber and shaft guide are kept close to the bottom of the shaft, extension guides may be used, as shown in the illustration, but under normal conditions a double crosshead will be better. If a skip or cage is used with the double crosshead, its guide shoes, when it is hoisted from the bottom of the shaft, should automatically engage the shaft guides. This is accomplished by the lower crosshead and the guide ropes, which insure that the cage and crosshead shall be parallel. Short blocks, tapered on three sides, as shown in Fig. 16, made from short sections of shaft guide, are bolted to the end of the guides, to facilitate the entry of the guides into the guide shoes. These short end pieces have been given the name, in the vernacular, of "pig snouts," on account of their shape. The term is expressive.

When used with a sinking cage, the lower crosshead (*C*) in Fig. 16 has the lower 6-in. channel extended 2 in. on each side beyond the edge of the guide shoe, so that it will strike on the block the cage will pass, and the releasing levers are straightened.

In those shafts in which a service cage is used, the ore being hoisted in skips, a sinking cage is particularly convenient for sinking below working levels, especially when used with a double crosshead. Under such conditions a pentice is left below the skips, and sinking is carried on through the cage compartment. The service cage is used as the upper crosshead, and the sinking cage is suspended below it by three ropes, the center rope carrying the sinking cage and the two outer serving as guides. (See Fig. 16.) They pass through holes in the lower crosshead, which runs on the shaft guides below the service cage. If the mine is working on day shift only, sinking is done at night by one or two shifts. The sinking cage is not detached from the service cage, with the use of which it does not interfere. In such event, if the shaft is deep, the rock hoisted is usually put off on one of the levels and is dumped into the storage pocket or, if the plat is high enough, it is dumped into mine cars from an overhead, temporary track. If the shaft is shallow, with sufficient headroom in the headframe, the rock will be hoisted to surface.

The double crosshead has completely replaced extension guides for sinking cages on the Marquette Range in Michigan, and, when it is better known, will probably do so in other districts.

Ordinarily, in shaft-sinking, a 6x19 regular lay hoisting rope is used, of cast-steel or plow-steel grade ("English acid" or "special acid"); but, in some districts, non-spin ropes are specified. These ropes have inner and outer layers laid up in opposite directions. Now that ropes with preformed wires and strands can be readily obtained, the special non-spin construction seems unnecessary.

Ropes may be attached to small buckets with safety hooks, or even "pigtails"; but for large buckets, skips, or cages, sockets are best, and clamps and thimbles are next choice. Not fewer than six clamps, preferably Crosby clips, should be used, the U-bolts always binding on the bitter end and not on the standing part of the rope. The factor of safety of the rope should be not less than 5, and 6 or 7 is better, and it should be frequently inspected for broken wires or undue wear.

Loading boxes have come into use within recent years for filling buckets, skips, and cars on cages. They are much like small quarry skips. Their advantages are that, on account of their low height, they can be easily filled, and they can be filled while the hoist is in motion. They are hoisted high enough to dump by small air hoists of 7 to 10 hp., mounted on the sinking stage or on the shaft timbers. If the capacity of each loading box is half that of the bucket, the bucket can be filled in a few seconds, and hoisting and shoveling can be carried on simultaneously. Loading boxes are particularly advantageous in conjunction with a sinking cage. A typical example of these boxes is shown in Fig. 18.

Mechanical loading in shafts has been practiced successfully in only a few instances, but with promise of better success in the future. The best opportunity is in large shafts. The Butler shovel with caterpillar tread has, I am told, been used for loading rock in a large shaft, with satisfactory results. The bucket was in the center of the shaft, and the shovel worked its way around it near the walls of the excavation. This shovel will dig its way downward on a slope of 30 per cent.

Recently, the Champion Copper Company, at Champion, Mich., has used a scraper successfully for loading rock in the shaft. The shaft is a 70-deg. incline, long and narrow in section. The bucket, at one end, is filled by means of a slide and a scraper of the hoe type, 30 in. wide and weighing about 650 lb. The scraper is hauled by a double-drum electric hoist, driven by a 15-hp. motor, which is mounted higher up in the shaft. The arrangement is shown in Fig. 19 (Inf. Circ. 6515 U. S. Bureau of Mines, by Albert Mendelsohn).

Clamshell and orange-peel buckets have been tried for loading rock in shafts, but have not been successful. They are often used, however, in drop shafts and in shafts sunk in sand, gravel, earth, or clay.

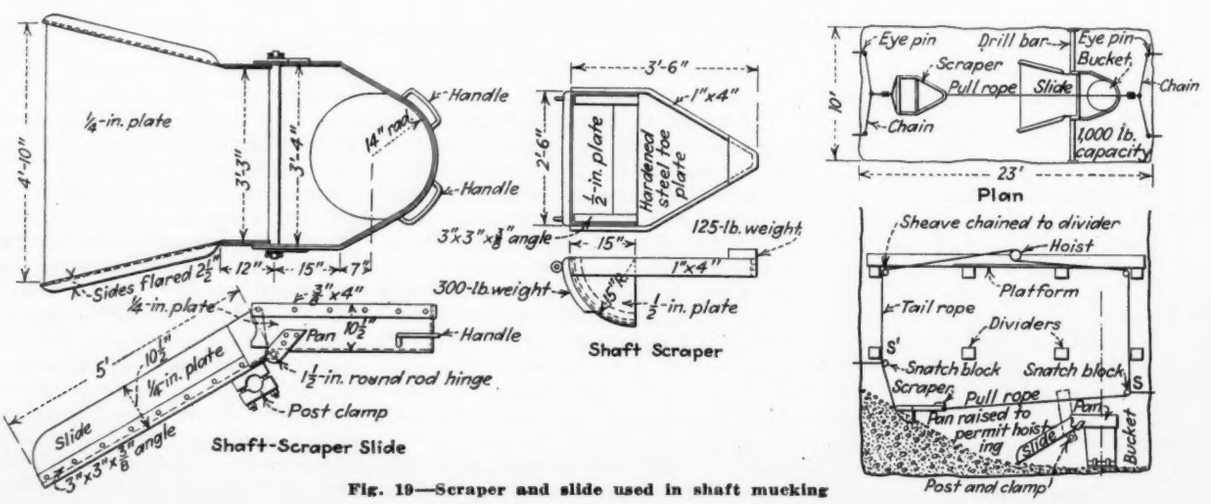
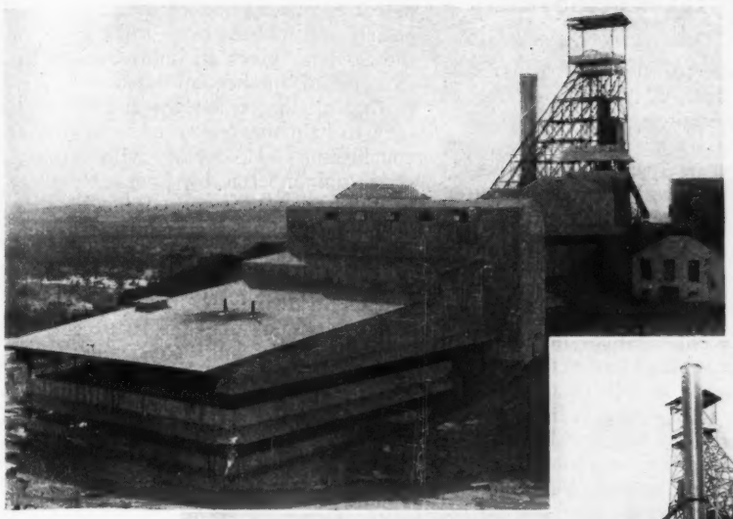
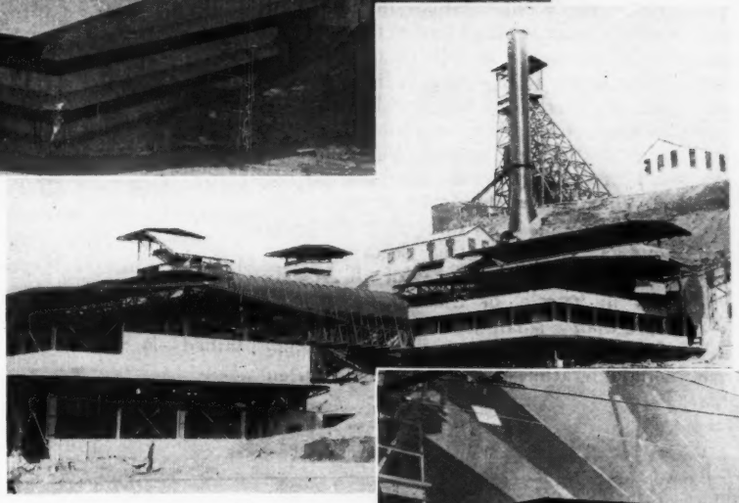


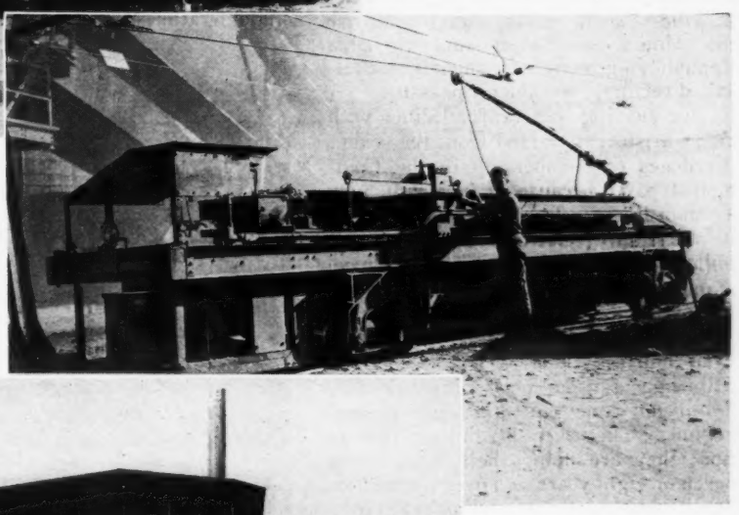
Fig. 19—Scraper and slide used in shaft mucking

At Mount Isa Mines

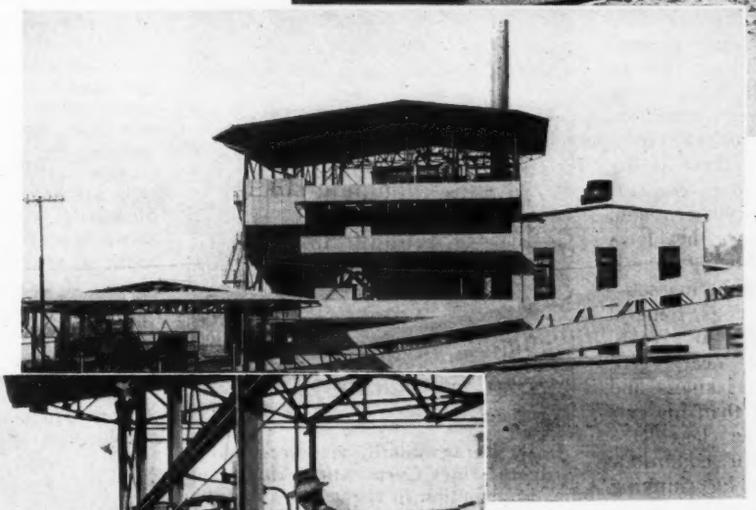




At the upper left is a picture of the Urquhart hoisting shaft and the 2,000-ton lead-silver concentrator of Mount Isa Mines, in northern Queensland, Australia. Production was started in May, 1931. The sintering plant, blast furnaces, and Cottrell plant of the lead smelter, which handles the concentrate, are shown in the second picture.

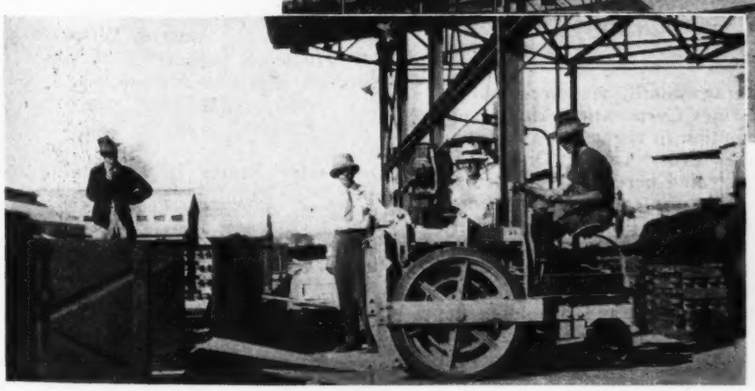


A charge car, with automatic weigher, used in the Mount Isa smelter is shown at the right. Difficulties encountered in handling the carbonate concentrate have retarded bringing the new smelter up to capacity, but it is scheduled to be producing 6,000 long tons of pig a month by the middle of 1932.



Two 5,000-kw. steam turbine generators are part of the equipment of the Mount Isa power house, shown at the right

These pictures were kindly furnished by Roland Blanchard, consulting geologist



Stacking lead bars in a railroad car for shipment to the coast at the Mount Isa smelter. M. J. Callow, who supervised smelter construction, is standing in the railroad car.

An American Engineer in Russia

The author of "Working for the Soviets"* needs no introduction to readers of a magazine that was privileged to publish in recent issues a series of articles by him on asbestos mining and milling in Canada, which are to be followed at an early date by a companion series on technical developments in the Russian asbestos industry. His first professional visit to the Union was made in 1929, at the time of the inauguration of the Five-Year Plan, followed by a lengthy stay in 1930-31, when it was in full operation. Mr. RuKeyser has therefore had the opportunity to observe the translation of policy into practice, as well as the many changes that have demonstrated a fallibility the Soviet authorities have been wise enough to acknowledge.

The book under review provides an interesting running commentary on the experiences of an American mining engineer in modern Russia, as well as a colorful panorama of the life of many of the racial units that comprise the Union of Soviet and Socialist Republics—government officials, technical directors, workmen, peasants. A chance meeting in a railroad diner with an ex-aristocrat, barred from the scanty privileges of membership in the Communist party because of his birth, now a subordinate official in constant fear of the future, discloses the prevailing philosophy of the dispossessed: "I have a job . . . nothing more," he explained to Mr. RuKeyser, "a fairly interesting job . . . it keeps me going ten, twelve, even fourteen hours a day. There is little hope for our class of people . . . I am not antagonistic to the present régime. They have a difficult row to hoe; they are doing their best. Only seven or eight years ago over five million souls perished in a single year from famine and disease; today there is order out of chaos, already one sees factories springing up everywhere and fields ripe with wheat; hears the blows of axe in forest. What is being done is being done for the good of the masses—the hundred or so million dogs, some of whom my grandfather used to give to his mistresses as a New Year's gift, whom he used to flog for laziness and who mattered to him less than his cattle. No, we former upper-class Russians brought our present conditions entirely on ourselves. The thinking ones among us saw the reaction in the air, knew the inevitable would happen. Now it has—we must make the best of it. We must help our present masters build a new and greater Russia, a new Russia for our children who will be the children of the proletariat. After all, the history of a nation is not measured in terms of five years, nor ten, nor twenty-five. It is measured only in terms of generations and centuries. The decadent Russia of the past is dead. The present is perhaps superficially worse for the few; but probably better, in the long run, for the majority."

Mr. RuKeyser, as might be expected, deals at length with the technical difficulties encountered in organizing the operations of the asbestos trust for efficient and economical production, but the few engineering details to be found in the book are merely of minor incidence in the telling of a swiftly moving tale of experiences that involved a continuous succession of obstacles to efficient direction and smooth functioning of an intricate combination of human and mechanical factors. No pretence is made to infer that the journeyings and sojourns described are typical of the experiences of all Americans in Russia, or that happenings in more than a small section of a vast area of territory and among an indescribable conglomeration of peoples have been recounted. Mr. RuKeyser has the competent engineer's quality of stating facts clearly and without bias. "Working for the Soviets" gives an impression of the exercise of intellectual honesty in disclosing all that is known and is necessary to help the reader to form correct conclusions. He writes with an easy style, unhampered by long experience in the preparation of engineering reports and technical accounts. He concludes the book with a confession of indecision and wonder as to what will happen in Russia, believing that "the furious tempo of the Plan can be psychologically and physically maintained only so long and no longer. It is a foot-race with a handsome prize for all the contestants at the finish. But there must be a finish—and there must be prizes for all. One cannot keep a mule walking forever by baiting him with a feed-bag. If eventually it is not given to him he will drop of sheer starvation. How long the Soviets can keep the masses 'pepped up' to a war psychology—how long this childish enthusiasm for a new toy can be maintained—is anybody's guess."

A. W. ALLEN.

*"Working for the Soviets." By Walter Arnold RuKeyser, New York: Covici, Fried, Inc., $3.

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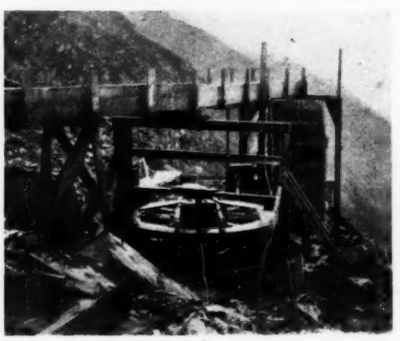

J. G. Baragwanath, vice-president of Pardners Mines Corporation, kindly supplies information in regard to a one-man mill in the Crow Creek District of Alaska, illustrated herewith. It consists of an overshot water wheel connected by chain drive to a three-stone arrastra equipped with a small amalgamation plate. The owner and operator, Henry Barneberg, shown in one of the pictures, does his own mining and development; he shares with his dog the task of packing the sorted ore a distance of about 2,000 ft. down a precipitous slope to the mill.

The Consumer Pays

To the Editor:

With reference to your editorial on the copper industry in your last issue, I wonder why something is not done by the producers to make copper more available to the final consumers. In illustration: A friend of mine had recently some roofing repairs made, and in the bill sent in by the roofer for this repair was an item for sheet copper at 35c. per pound. Inquiry of a metal dealer here in Boston revealed that this dealer was charging from 20c. to 21c. per pound for large lots, and 25c. per pound for small retail lots. There seems to me a large discrepancy between this price that the consumer has to pay and the 6c. or 7c. per pound for ingot copper. I have no doubt that such practices are prevalent elsewhere as well as in Boston. I know that people refrain from using copper in repair and construction work on account of the high price charged them. It seems to me that the producers of copper should do something to improve this situation and thereby increase the demand for their product, both for their own benefit as well as the consumer's.

ARTHUR WINSLOW.

Boston, Mass.

■

A. Chester Beatty is vacationing in Egypt.

P. R. Hines has been elected president of the Oregon Technical Council.

H. R. Plate has removed his office to 85 Second St., San Francisco, Calif.

Hugh R. Van Wagenen has returned to Los Angeles from Oaxaca, Mexico.

R. O. Speed, dredge engineer with Siamese Tin Syndicate, London, is in Australia.

Felix Burns, mining engineer with Electrolytic Zinc, is on loan to Mount Coolon Gold Mines.

Lucien Eaton sailed from New York for England on Feb. 19, expecting to return in about three months.

Alan Greenless, until recently superintendent of roasting plants of Electrolytic Zinc, is now with Mount Isa Mines.

D. D. Wessels, of Detroit, has been named president of Canusa Mining & Exploration, to succeed **George Miller.**

Herbert C. Enos announces removal of his office from the Hollingsworth Building, Los Angeles, to Chatsworth, Calif.

A. J. McNab, of Rhokana Corporation, recently left London on a visit to the company's properties in Northern Rhodesia.

J. Malcolm Newman, formerly managing director of Alluvial Tin (Malaya), has joined the board of directors of Sydney Siam Tin.

Klem B. Gross, formerly on the metallurgical staff of the Electrolytic Zinc at Rosebery, Tasmania, has joined Mount Isa Mines, Queensland.

Martin Schwitter is now in practice as an independent chemist and assayer, with headquarters at 483 Raymond Boulevard, Newark, N. J.

W. J. Pike sails from San Francisco on March 11 for the Philippine Islands, where he will join the technical staff of Benguet Consolidated Mining.

R. N. Pursel, until recently mining engineer with Cia. Aramayo de Mines en Bolivia, at Quechisla, Bolivia, can be reached at his residence in Danville, Pa.

R. W. Bender and **T. Lewis** have been appointed to assist the California State Tax Research Bureau in making an appraisal of mining property in California.

T. L. H. Shone, of Postmasburg, South Africa, discoverer of the Postmasburg manganese fields, has joined the directorate of South African Manganese, Ltd.

George A. Kervin, at one time manager of the Mason Valley Mines Company, of Nevada, is now superintendent of the Murchie mine, near Nevada City, Calif.

Joseph E. Pogue, Philip W. Henry, Prof. Robert Peele, Ralph R. Sweetser, and **A. B. Parsons** have been elected to the Board of the Engineering Societies Library.

C. E. Wilson, for nine years general manager of Worthington Pump & Machinery Corporation, has been appointed vice-president in charge of industrial relations.

Archibald Lawrence, formerly of Howey Gold Mine, western Ontario, has been appointed manager of the Parkhill mine, at Michipicoten, Ont., succeeding **J. A. Roussac.**

Howard D. Smith, president of Consolidated Coppermines, was married in New York recently to Mme. Cyrena Van Gordon, mezzo-soprano of the Chicago Civic Opera Company.

Frederick R. Weekes and **E. Perry Crawford** have taken over the Yankee Girl gold mine, at Ymir, British Columbia. They will continue their consulting practice at 475 Howe St., Vancouver, B. C.

George H. Dormer, safety head of the Virginia district operations of Oliver Iron Mining, has been promoted to superintendent of mines at Ishpeming, Mich., to succeed **Capt. F. E. Keese,** retired.

Dr. Victor Dolmage is the new president of the British Columbia Chamber of Mines. Other officers recently elected include **G. S. Eldridge, H. P. McCraney,** vice-presidents; and **W. Mason Rooke,** honorary treasurer.

A. J. Keast has been appointed manager at the Beattie Gold Mines, in Quebec. The position from which he resigned on the staff of Macassa Mines, Ltd., at Kirkland Lake, Ont., has been assumed by **G. A. Howes.**

William Morris Davis, emeritus professor of geology at Harvard University, has been awarded the Penrose Medal by the Geological Society of America, established in 1927 by the late R. A. F. Penrose, Jr.

E. C. Andrews, Chief Government Geologist of New South Wales Mines Department, is retiring, and will prepare a treatise on the geology of Australia. Three years ago he delivered the Silliman lectures at Yale University.

A. L. Russell, traffic manager of National Exploration Corporation, of Prescott, Ariz., was recently elected vice-president of Corporate Mining Trust. Mr. Russell will continue his duties with National Exploration for the present.

L. K. Requa was recently elected chairman of the San Francisco Section of the A.I.M.E.; **Wendel C. Hammon,** vice-chairman; **James G. Parmelee,** secretary-treasurer; and **Fred W. Varney,** member of the executive committee, all to serve during 1932.

L. J. Walters, who contributes the article on the handling of diatomite in this issue, is a graduate in mechanical and structural engineering of the City of London College. Varied experience as a designer and draftsman preceded appointment on the staff of H. Kenyon Burch in 1921, in connection with the designing of the Copper Queen, Nacozari, Old Dominion, Burro Mountain, and Ajo concentrators. In 1924, Mr. Walters was appointed chief mechanical engineer to the Celite Corporation, largest miners and processers of diatomite in the world, operating at Lompoc, Calif.; and in 1929 he was appointed assistant designing engineer in the work of adding new units to the Ajo concentrator. Followed appointment as chief engineer to the Dicalite Company, Palos Verdes, Calif., in 1930, and other professional work as consultant.

Edwin E. Graupner, Butte, Mont., writes that renewed efforts are being made to induce Congress to exempt miners and prospectors from the necessity of doing assessment work on unpatented mining claims for the fiscal year 1931-1932. He points out that the depression makes it almost impossible for the individual miner or prospector to raise the necessary "grubstake" for doing the required assessment work on base-metal claims. Mr. Graupner feels that the chief objection to the exemption measure (that large companies would take advantage of the opportunity to allow necessary development on their claims to lapse, thus reducing employment) is not valid. The number of claims held by large companies is comparatively small, and most of them will do the work in spite of the removal of compulsion. From Los Angeles, in California, comes word of petitions to Senators and Representatives, for the same purpose, on the initiative of **John F. Duling** and associates.

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Obituary

H. L. J. Warren; at Brookline, Mass., Feb. 10; aged 79. Long identified with mining in the West.

H. H. Williams; at Seattle, Feb. 6; aged 68. Prominent in mining industry of Alaska and British Columbia.

W. W. Tiner; at San Francisco, Feb. 22. Engaged in mining in many parts of the world in association with Robert Hutchens, of London.

G. W. Beck, Jr., head of Beck Mining & Royalties Company; at Miami, Okla.; aged 48. Prominently connected with mining in Tri-State district for seventeen years.

Paul Truman Boise, at Salt Lake City, Utah, Feb. 23; aged 46. Mr. Boise had been active for many years in promoting mining enterprises in the Park City and Tintic districts, as well as in Idaho and Nevada.

Burrows Sloan, chairman of board of General Refractories, at Philadelphia, Feb. 28; aged 55. A graduate from Princeton, Mr. Sloan became associated with the Reading Company and the Harbison-Walker Refractories Company. He organized General Refractories in 1911.

J. B. Francis Herreshoff, honorary vice-president of Nichols Copper Company and past president of American Chemical Society, died at his home in Atlanta, Ga., on Jan. 30, at the age of 81. Born in Bristol, R. I., he studied at Brown University, where he also taught analytical chemistry until in 1870 he became associated with G. H. Nichols & Company. For outstanding work in chemistry and metallurgy he became the first recipient of the Perkins Medal. Best known among his many inventions is the roasting furnace bearing his name, which made possible the treatment of fine sulphide ores, formerly of little commercial value, thus opening a new era in roasting technique.

Improvements in Mineral Scrubbing

J. K. Towers
Division Engineer
Hardinge Company, York, Pa.

THE interesting article on hydraulic classification of fine phosphate in Florida, by R. G. O'Meara, in the Dec. 28, 1931, issue of *E.&M.J.*, prompts me to supplement the information with data on the release of phosphate from a clay matrix. The flowsheet accompanying that article shows a Hardinge mill for handling the product between two sets of double sizing trommels. At another property the clay in which the pebble is imbedded is extremely tough, and the process applied is shown in the flowsheet below.

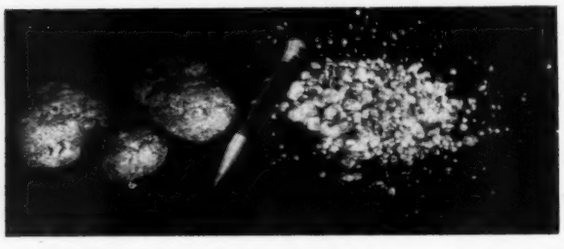
The phosphate companies in recent years have learned to look to the experience of the mining industry. One application is indicated by their use of a modification of the Hardinge ball mill for the reduction of clay balls and the release of clean phosphate. Attrition in grinding is nothing more than a rubbing of each particle against another surface or particle. However, attrition with resultant diminishing in size of the particle so the scrubbing must be terminated before actual reduction in pebble size, with resultant losses of phosphate.

Because of the soft nature of the pebble, losses will result because of attrition brought about during extensive screening and cleaning operations. These losses have been materially reduced as a result of the installation of Hardinge scrubbers in place of log washers. The hydraulic classification of the fine material referred to by Mr. O'Meara has been possible because of the greater release of clay from phosphate.

Another interesting procedure refers to the separation of the heavy flint that frequently occurs with the mass, as mined. This is accomplished directly in the scrubber, the weight of this material being so much greater than either the pebble or the clay. An accumulation takes place, and at intervals of about once a week the accumulated flint is emptied through the manhole.

Coincident with the work done with the Hardinge conical scrubber in the phosphate field, I have come in contact with somewhat similar work in the sand and gravel industries, the crushed-stone industry, and others, where problems are somewhat similar. In certain deposits of sand and gravel appears heavier flint and slag, in addition to clay. All this requires treatment of a different nature than has been followed in the past by companies that are now required to meet state specifications with respect to the amount of clay and other foreign matter in sand and gravel. In some states the requirements are such that the washing of crushed stone is necessary. In this field, the foreign material rarely amounts to more than 20 per cent. Frequently less than 5 per cent clay will be found in the mass. Nevertheless, these states seem justified in requiring an absolutely clean stone.

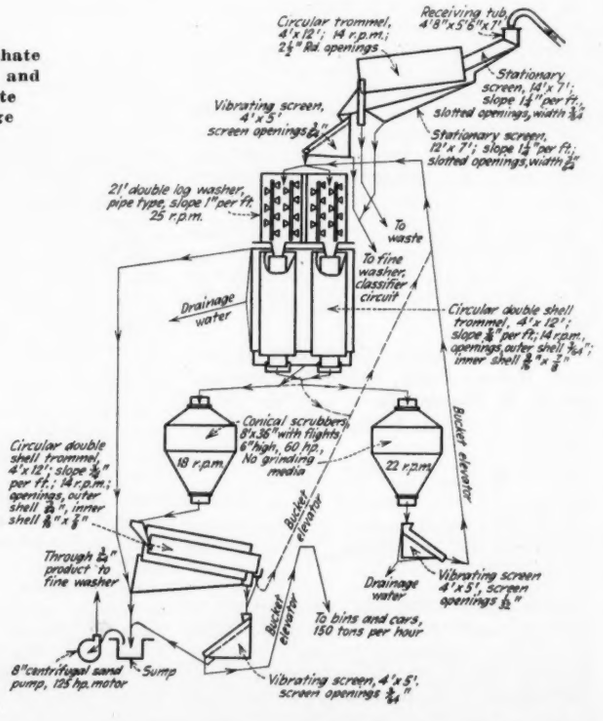
A noteworthy application of the conical scrubber in this work is an installation in New York State, where a 6 ft. x 48 in. conical scrubber of the latest design readily produces 80 tons per hour of almost clay-free product. This resulted in the addition of four 8 ft. x 48 in. scrubbers for installation in several of the other plants of this company.



Clay-phosphate feed (left) and phosphate discharge

is not ordinarily wanted in the cleaning of phosphate, although this is true only as it affects the phosphate and not the clay that surrounds each phosphate particle. An important factor, therefore, is the degree of attrition of scrubbing.

Fortunately, the conical scrubber lends readily to the rapid circulation as well as a quick discharge of the mass. Therefore, the degree of attrition can be adjusted to suit almost any requirement by varying the rate of feed. The effects of this action in the scrubber are illustrated in the photographic reproduction presented herewith, showing the large clay-ball feed to the scrubber, and the clean phosphate released from the clay while passing through the scrubber. The phosphate must be freed from clay,


Circular trommel, 4'x12'; 14 r.p.m.; 2½" Rd. openings
Receiving tub, 4'8"x5'6"x7'
Stationary screen, 14'x7'; slope 1½" per ft., slotted openings, width ¾"
Vibrating screen, 4'x5' screen openings 2½"
Stationary screen, 12'x7'; slope 1½" per ft.; slotted openings, width ¾"
21' double log washer, pipe type, slope 1" per ft. 25 r.p.m.
To waste
To fine washer, classifier circuit
Drainage water
Circular double shell trommel, 4'x12'; slope ¾" per ft; 14 r.p.m., openings outer shell 7/64"; inner shell ⅜"x ⅜"
Conical scrubbers, 8'x36" with flights, 6" high, 60 hp., No grinding media
18 r.p.m.
22 r.p.m.
Bucket elevator
Circular double shell trommel, 4'x12'; slope ¾" per ft.; 14 r.p.m.; openings, outer shell 2½", inner shell ⅜"x ⅜"
Through ½" product to fine washer
Bucket elevator
Drainage water
Vibrating screen 4'x5', screen openings ½"
Bucket elevator
To bins and cars, 150 tons per hour
8" centrifugal sand pump, 125 hp. motor
Sump
Vibrating screen, 4'x5', screen openings ¾"

Flowsheet of phosphate rock scrubbing plant

Safety Device for Cages That Cannot Release Accidentally

J. W. Scherrer,
Maastricht, Holland

CAGE-ARRESTING devices in their action usually depend upon changes in the tension of the hoisting rope, a condition which in practice has been found to possess two disadvantages. First, not all changes in the rope tension should bring the arresters into action; second, the spring involved cannot be made sufficiently strong to force

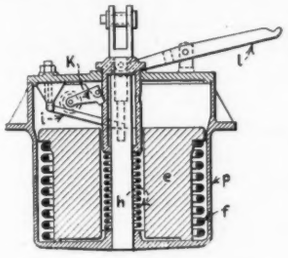


Fig. 1—Schönfeld cage-arresting apparatus

the gripping device against the guide in all cases of rope breakage. The first of these facts is commonly known; the second may need further elucidation.

The strength of the spring is limited for practical reasons. During hoisting, the motion of the cage does not depend solely on the motion of the engine; the resistance in the guides and other influences give the cage a separate up-and-down motion, easily felt when the engine is slowing down prior to stopping at the bottom of the shaft. This motion, accelerating or retarding, as the condition may be, necessarily affects the rope tension. Repeated measurements of this acceleration and retardation have shown it to have a maximum of about 7 m./sec.[2] If the spring operating the safety catches has to sustain the entire weight of the empty cage, the slightest retardation on the down trip would bring the catches into action. To prevent this the cage must be overweight—that is, the spring should carry only a part of the weight of the empty cage. This portion carried by the spring must be determined by experiments.

As acceleration due to gravity is about 10 m./sec.[2], a retardation of 7 m./sec.[2] caused by guide resistance will reduce the rope tension 70 per cent, leaving 30 per cent of the weight of the cage as the maximum load which the spring must be able to carry without bringing the safety catches into action. In practice, the strength of the spring is limited to 25 to 33 per cent of the weight of the empty cage. This limitation of the spring makes its action uncertain in case of rope failure. In such event, the spring has still another function to perform: that of pulling the end of the rope attached to the cage in the direction of the falling cage, giving the rope end an acceleration exceeding that of the cage. This piece of rope may offer sufficient resistance to prevent the safety catches from getting into action, which is one of the reasons why cage arresters should never be wholly relied upon.

Recently, another device has been developed, patented by Schönfeld in Germany. This apparatus, shown in Fig. 1, is placed in the cage. It consists of a cast-iron receptacle, containing an iron weight, partly supported by a spring. If the rope breaks, the cage will start to fall at a rate of acceleration depending upon the resistance of the guides; as the weight no longer exerts any pressure on the bottom of the receptacle, it will be raised by the action of the spring *f*. Lever *i* then pushes lever *k*, releasing the strong spring *h*, whereby the safety catches are set free.

Accelerating and retarding forces during hoisting operations naturally affect the pressure exerted by the weight on the bottom of the receptacle. To prevent accidental release the strength of the spring must be limited. The same limitations characterizing the type of cage arrester first discussed are also found in this case, necessitating the use of a spring strong enough to carry only 30 per cent of the weight *e*. This means that the device will not function until the acceleration has reached 7 m./sec.[2]. Unfortunately, this acceleration is not always attained when the cage is falling down the shaft. Even though the device works independently of the rope tension it therefore does not afford complete protection. For this reason a new instrument, based on the same principle, was developed by the Dutch Bureau of Mines. The action of this instrument does not depend upon the degree of acceleration only, but also upon its duration.

As the minimum acceleration of a falling cage is known to be 5 m./sec.[2], the spring must be sufficiently strong to support half of the weight; during hoisting operations accelerations and retardations exceeding 5 m./sec.[2] are, however, possible, causing the weight in the receptacle to move up and down. This weight, in the Schönfeld apparatus, must have a mass of 100 kg. to develop sufficient force to prevent danger.

From measurements a maximum duration of half a second has been found for acceleration and retardation of the cage during hoisting, except when caused by motion of the hoisting engine, which never reach a value of 5 m./sec.[2]. The distance which the weight in the receptacle is raised may therefore easily be determined.

Assuming, for instance, a downward acceleration of 7 m./sec.[2] for 0.5 sec., the pressure of the weight against the bottom of the receptacle is reduced by seven-tenths. As the spring is designed to sustain half of the load, the weight is driven upward with a force of five-tenths minus three-tenths, or two-tenths its mass, giving it an acceleration of $0.2g$ during 0.5 sec. Applying the formula $s = \frac{1}{2}at^2$, the distance the weight is raised is $s = \frac{1}{2} \times 0.2 \times 10 \times \frac{1}{2}^2 = 0.25$ m. To raise a weight of 100 kg. this distance is of course out of the question.

In the new Dutch device, shown in Figs. 2, 3, and 4, the weight, as well as the distance it is raised, has been reduced. The spring is made as strong as the Schönfeld apparatus; when the cage is falling the spring is released from the weight, permitting it free action, and making it absolutely reliable. The weight *e* is placed in a receptacle *b*, in which it can move vertically; it is connected to lever *g* by a lever *d* and the axle *f*. Spring *p* supports the piston *o*, which moves vertically in the cylinder *n*, and which is connected

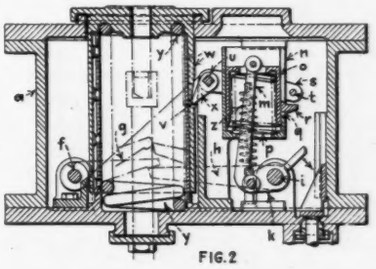


FIG.2

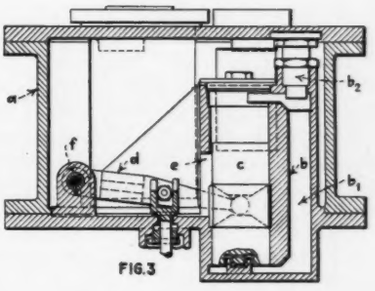


FIG.3

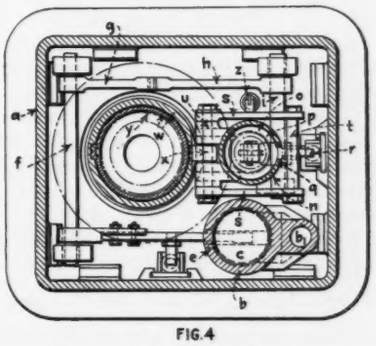


FIG.4

Figs. 2, 3, and 4—Cage-arresting device developed by Dutch Bureau of Mines

to lever *k* by a pole *m;* lever *k* in turn is connected to lever *h* by the axle *i*.

As seen in Fig. 2, lever *g* is supported by lever *h*, and the weight *c* is carried indirectly on spring *p*. The lengths of levers *k*, *h*, *q*, and *d* are chosen to make the strength of the spring such as to sustain ten times the load developed by the weight *c* on lever *d*. This combination of a strong spring and a light weight makes possible a reduction in the mass of the whole apparatus.

As the weight has a mass of only 5 kg., and the spring can sustain a load of 25 kg., the weight *c* will move upward in the receptacle when the downward acceleration of the cage is at least 5 m./sec.^{2}, in which case the pressure exerted on the bottom of the receptacle is reduced to 2.5 kg. The cast-iron box as well as the space above and below the cylindrical weight *c* are completely filled with oil; the channel *b*, between the upper and lower space, may be wholly or partly restricted by the screw b_1. In this way the motion of the weight may be suppressed at will; whenever the motion exceeds a certain limit lever *g* is released from lever *h*, causing the weight *c* to drop down.

Spring *p* is then free to perform its function with absolute certainty; as it expands, the arm *r* releases bolt *t*, which in turn acts upon lever *s*, removing arm *x* from cylinder *w*. This releases spring *y*, which presses the gripping device against the guides. In this manner the apparatus is brought into action only when the acceleration of the cage on the way down, or the retardation on the way up, exceeds a definite value over a certain period of time. As this can take place only when the cage is no longer attached to the rope, the apparatus cannot release the catches accidentally. It also affords complete protection in case of danger.

■ ■ ■

Novel Propeller-Type Mine Fan Used at Butte Excites Attention

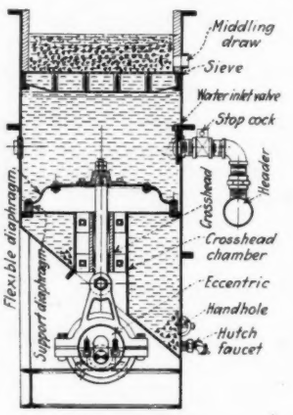
Other Recently Introduced Equipment

A propeller-type mine fan possessing very definite advantages and having only two blades which are set on a pitch and together cover about 230 deg. of arc, thereby differing from fans having a multiplicity of airplane-type propellers, was described at the recent meeting of the American Institute of Mining and Metallurgical Engineers, in New York, in a paper presented by A. S. Richardson, ventilation engineer for the Anaconda Copper Mining Company. This fan is made by the Westinghouse Electric & Manufacturing Company and is in service at the Moose shaft of the Badger mine, in Butte. It develops an efficiency of 83.5 per cent, costs less, when completely installed, than other fans, according to Mr. Richardson; can be easily reversed in one minute without entering the fan drift to throw doors, and will give any water gage desired up to 40 in. In service at the Moose shaft it produces 290,000 cu.ft. per minute at 9.8-in. water gage. The atmospheric pressure at the collar of this shaft is 23½ in. The fan takes 565 hp. and is driven by a 700-hp. motor. It is 72 in. in diameter and the peripheral speed of the blades is 550 ft. per second. The hub is one-third the diameter. Because of escaping sulphur fumes the fan duct is lined with lead. The fan appears to be applicable for main, booster, and auxiliary ventilation service. It has also been used in central station work to cool a generator and by the U. S. Navy for ventilating warships.

Diaphragm Jig Efficient and Compact

During the past five years a jig known as the Bendelari jig has been developed in the Tri-State zinc-and-lead district. Ninety-eight cells of this type will be installed in the new Central mill of Eagle-Picher Mining & Smelting Company. The cell is so designed that it can be adapted for use with any mineral that can be jigged. Its compactness, light weight when loaded, and low headroom requirement make it particularly desirable for work on dredges.

The distinctive characteristics of this jig, which is shown in the accompanying cut, are its efficiency and compactness, according to F. N. Bendelari. A diaphragm in the pulsation chamber takes the place of the usual plunger cell. A tube through the hutch permits of a connection between the diaphragm head and the eccentric without using a stuffing box. The use of a wrist pin and crosshead produces a vertical action of the diaphragm. The corners of the plate which carries the diaphragm are cut away, permitting fine material to settle in the still water in the hutch. Water from a manifold is admitted below the sieves through check valves on the down stroke of each diaphragm. Water so admitted is regulated in amount by a control valve on each cell. By operating these valves the action of the jig can be changed while the jig is running.



Sectional drawing of diaphragm jig

The action of the jig is uniform all over the bed. It is positive, inasmuch as there is no plunger slip. Stratification is very rapid. With the higher speeds possible, very fine material can be efficiently jigged.

By reason of its compact design, the jig occupies less than half the space ususally required for jigs. Foundations are less expensive. The jig may be operated on the mill floor. Being compact, it can be moved through mill doors.

Improved Magnesite Brick Produced by New Process

Research work conducted in the laboratory of the General Refractories Company, of Philadelphia, Pa., in the last few years, has borne fruit in the successful development of an unburned magnesite brick of sufficiently superior qualities to warrant its application in metallurgical work. One smelting company is said to have decided to use this brick in all work at its largest copper refinery, which incidentally is also the world's largest. The superintendent of another copper refinery states that tests with the brick have been successful, and that the new product appears to be superior. An engineer engaged in nickel smelting in the Sudbury district is likewise reported to be interested in the possibility of introducing the brick in his plant, where at present large quantities of chrome brick are used. Some tests are said to have been planned at this plant.

The research work referred to in the opening sentence was undertaken because the unburned magnesite brick previously made were giving unsatisfactory results, principally by reason of shrinkage in service. Preliminary study showed this shrinkage to be due in the main to inadequate methods of pressing

employed in making the brick. To remedy this, the method of manufacture was improved by increasing the pressure tenfold; by grading the sizes of the magnesite particles so that, when mixed, the percentage of voids would be reduced; and by coating the particles with a colloid which not only served to lubricate them so that they would fit together better under pressure but would also act as a bond between them after drying and sintering.

To make the new-process brick called for the development of a new press capable of developing the requisite high pressure and the high speed necessary for economical manufacture. Likewise, special screen equipment was developed to give the grain sizes found most suitable. Proper proportioning of the different sizes had been worked out in the laboratory previously. Manufacturing operations under the new process were conducted under strict laboratory control.

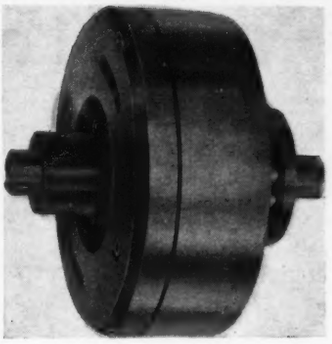
Tests of the new and the old brick showed that the unburned product was superior to the burned in the percentage of shrinkage in service and also in crushing strength, and that, in addition, it was better bonded at high temperatures. They also demonstrated the unburned brick to be superior to the burned in the matter of spalling. Under the new process the unburned brick were also found less permeable to gases than the burned brick, so that with the former the leakage of a gas through a furnace wall would probably be considerably reduced.

The new method of making the brick gave a lower production cost. The new brick, moreover, are pressed exactly to size without shrinkage so that they can be laid with better joints in furnace walls, this in turn reducing the likelihood of slag and metal penetration.

As indicated by the comment in the first paragraph, the results in service have substantiated the laboratory findings. They have demonstrated that the unburned magnesite brick made by the new process can be used to replace burned magnesite and chrome brick in general, and also fireclay and silica brick to a considerable extent.

Two Magnetic Devices—A Clutch and a Clutch-Coupling

Two new products have recently been turned out by the Dings Magnetic Separator Company, Milwaukee, Wis. One is a magnetic-friction type clutch developed to meet the demand for powerful clutches of smaller physical dimensions. This is a simple clutch having contact faces on both sides of a spring disk which are squeezed between the magnet and armature elements when the coil is energized. Although the power transmission of this clutch is double that of a single friction contact face, actually it is of approximately 75 per cent greater torque with the same magnet strength, a material advantage within certain physical limits. Provi-



Externally, the new magnetic-friction clutch and the magnetic clutch coupling appear the same

sions are made for adjustment to compensate for wear of the contact faces. The clutch is made with and without ball bearings between the hubs of the two clutch members, depending upon installation conditions. The company has also developed a single spring disk clutch following the same general construction as that described but having three friction contact faces with the consequent increased torque and power transmission.

The other of the two devices mentioned is a magnetic clutch coupling having serrated contact faces, this to provide an absolutely fixed clutch the design of which obviates the danger of permitting the clutch contact faces to slip or creep. The claim is made that when this clutch-coupling is energized, such creeping will not occur and that the adjustments will remain fixed within the torque limits of the clutch. These, it is said, are exceptionally high. Up to the present, all installations of this clutch-coupling have been in connection with rolling mill screw-downs. However, the manufacturer considers its field of use to include any application where synchronization of the speeds of two power drives is required.

A New Drill Sharpener

A drill steel sharpener capable of handling any size of hexagon, quarter-octagon, or round steel, and that can forge and sharpen double-taper bits up to 3-in. on steel up to and including 1½-in. round, has just been introduced by the Gardner-Denver Company, Quincy, Ill. This is the Model DS6. It incorporates a new scale blower operated by a small lever on the right side of the machine within easy reach of the operator. Over-all height is 48¼ in., weight 2,000 lb., and the diameter of the round base 28 in.

Timkenizing of Locomotive Successful

Beneficial results were obtained by the Timken Roller Bearing Company from its 21 months' test of a locomotive equipped with anti-friction bearings. This test was begun in April, 1930, since which date the locomotive has run 100,000 miles. During the test period no instance of bearing trouble was recorded nor any train delay that was occasioned by bearing disability. Possibilities for lower maintenance were fully demonstrated in the substantial savings effected by obviating repair or replacement of the bearings themselves over long periods of operation. Increase in lubrication efficiency further contributed to lower maintenance costs. The company reports that attention to lubrication was found unnecessary over a period of several months at a time. Moreover, the tests indicate a decided increase in starting efficiency. Ample demonstration was given that the bearings increase the tractive effort upon starting trains. The statement is made that, unassisted, the locomotive with these bearings started trains that ordinarily would require a helper to get moving. This superior starting ability was noticed not only on level tracks but on heavy grades also, and is ascribed to the virtual elimination of frictional resistance in the axle bearing.

For Mine Ventilation

A flexible ventilating pipe, called "Mine-Vent," for use in mines, has been introduced by American Brattice Cloth Company, Warsaw, Ind., and H. J. Hersey & Company, Salt Lake City. A feature of its design is the coupling, which is said to facilitate joining lengths, to allow use of odd lengths without sewing, to permit shortening the line readily; to eliminate twisting; and to allow slight angles to be made. The pipe may be had in 15-, 25-, 50-, 75-, and 100-ft. lengths and in diameters 8, 12, and 16 in. It is hung from eyelets.

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BULLETINS

Mine Safety—Mine Safety Appliances Co., Braddock, Thomas and Meade Sts., Pittsburgh, Pa. Bulletin No. 96, Edison permissible electric hand lamps and portable searchlight.

Geared Motors—Westinghouse Electric & Manufacturing Co., Nuttall Works, Pittsburgh, Pa. Leaflet No. 20536, "Gearmotors," consisting of speed reducers combined with induction motors.

Centrifugal Pumps—Allis-Chalmers Manufacturing Co., Milwaukee, Wis. Bulletin No. 1642B, Type "M" double-suction multi-stage pump.

Compressed Air—Sullivan Machinery Co., 30 Church St., N. Y. City. Bulletin No. 100-C, tanner tanks and tanner-gas anti-freeze air-line system.

Power Transmission—Chain Belt Co., 1600 West Bruce St., Milwaukee, Wis. Catalog No. 440, roller chains and sprockets.

Vibrating Screens—Universal Vibrating Screen Co., Racine, Wis. Catalog No. 99.

Shafts and Tunnels—The Dravo Contracting Co., Pittsburgh, Pa. Bulletin No. 302, shafts, slopes, and tunnels; also Bulletin No. 301, on cementation.

Steam Generators—General Electric Co., Schenectady, N. Y. Bulletin No. GEA-1372A, supersedes No. 1372, electric steam generators.

NEWS OF THE INDUSTRY

UNITED STATES

San Francisco, Calif., Feb. 24

Lincoln Gold on Mother Lode. Lincoln Gold Mining has filed articles of incorporation at Jackson, Calif., Included in the holdings are the Lincoln, Mahoney and Wildman gold properties on the Mother Lode near Sutter Creek. Gordon R. Campbell, former head of Calumet & Arizona, is president of the new company.

Thies Builds Gold Mill. The "Thies Organization" has started the installation of a 100-ton mill at the Shasta View gold mining claims, near Shasta out of Redding, Calif. Some twenty mining claims are in the group, and exploration work has been in progress.

Yellow Aster Mill Running. In the Randsburg district about 30 leasers are supplying gold ore to 10 stamps of the Yellow Aster mill and 40 stamps are being operated upon ore from other parts of the Yellow Aster property. Placer operations are engaging some attention.

Idaho-Maryland Reopens Brunswick. Idaho-Maryland has placed a new ball mill in operation at its Brunswick mill in the Grass Valley area. Idaho-Maryland is now milling more than 150 tons per day and is preparing to reopen its Brunswick mine. The company recently purchased the 240-acre Loma Rica tract adjoining its mining property.

Wallace, Idaho, Feb. 27

Sunshine Stands Alone. The Sunshine mine, in the Coeur d'Alene district, has the distinction of being the only mine in the United States producing silver as the only source of profit which maintained continuous and normal production throughout 1931. The profit was modest, the company paying a dividend of 3c. per share in the last quarter. During the year, the mine was operated six days a week and the mill seven. Production was 183,441 tons of ore running 14.368 oz. silver per ton, 0.0032 per cent lead, and 0.0012 per cent copper. Milling loss was 1.24 oz. silver per ton with a trace each of lead and copper. Mining cost was $2.5963 per ton, milling cost $0.3888, a total of $3.6357. In December, owing to the persistent low price of silver, the force was reduced from 250 men to 175 and another reduction will probably be made. Only ore paying at the present low price will be mined. Daily tonnage will be about 300 compared with over 500 in 1931.

Spokane, Wash., Feb. 23

Jack Waite Ships. The Jack Waite property shipped 22 carloads to the Bunker Hill smelter in January. Production during the month is reported to have exceeded shipments because of bad weather, which for ten days prevented the trucks from making a 4-mile haul to the railroad. High-grade ore, revealed seven months ago, is reported to persist on the 200-ft. level, where drifting is under way. The greater part of the production was from this level and almost all of it was broken in development work.

Spring Hill to Reopen. Arrangements are under way for a resumption of operations at the Spring Hill property of the Montana Mines Corporation. Sufficient ore is said to be in the chutes and broken to keep the mill supplied for six months, at the rate of 250 tons daily, A campaign of development is proposed, with drifting on the 500-ft. level along the strike of the contact in which the orebodies are found.

Butte, Mont. Feb. 20

East Helena Resumes. On Feb. 15 the East Helena smelter of the American Smelting & Refining resumed operations with 100 employees.

Salt Lake City, Utah, Feb. 24

Yankee Output Up. Yankee Mines, owned by American Smelting & Refining, will increase production from its American Fork property during March from its present rate of 1,500 tons monthly to 2,000 tons. Later in the spring, when weather conditions improve, it will be raised to 3,000 tons monthly. Ore shipments are averaging around $10 in gold and 2 oz. of silver to the ton; and 2 per cent copper. The orebody has been opened up for a length of more than 600 ft., a maximum width of 60 ft., and a thickness of from 10 to 50 ft. Construction of a 4-mile cable tramway has allowed the company to maintain shipments during the winter, when ordinarily the roads are closed by the snow.

Bingham Properties Sold. Two mining transactions were consummated in the Bingham district of Utah during February, which should have an important effect on future activities. U. S. Smelting, Refining & Mining, according to D. D. Muir, Jr., manager, increased its Bingham acreage by buying from J. A. Hogle, Salt Lake broker and financier, the majority interest in the Bingham Prospect. Two years ago, the company purchased a minority interest from J. J. Beeson, Salt Lake geologist. The mine has been a considerable producer of lead and silver, making a net profit of $364,000 in 1929. The property, lying in the eastern end of the district, rounds out the huge block of ground owned by the smelting company at Bingham.

National Lead, through its subsidiary, Combined Metals Reduction, has acquired the Lavagnino group, including the first 8,000 ft. of the Butterfield tunnel, the Allread Kelley group, the Rob Roy, and other groups in the southeastern part of the Bingham district, southeast of the U. S. mine and the Utah Copper. Acquisition of this property completes the movement to acquire Bingham holdings begun last summer, when the company took over control of Park Bingham. The Butterfield tunnel, which extends into the property 12,000 ft., has its portal in Butterfield Canyon, about 3 miles from the railroad station at Lark.

Chief Consolidated to Curtail. Three Chief Consolidated properties will be closed on March 25. These are the Chief Con. No. 1 mine, the Grand Central, and the Plutus. Leasers will continue to work in the Eureka Lilly, and the company will continue operations in the Apex Standard, where the shaft is being put in condition to sink from the 900 to the 1,100 level to prospect for a condition similar to that in the adjoining Eureka Standard. J. Fred Johnson, general superintendent of the company's Eureka operations, will go to California to take charge of work in Chief Con.'s Placer County gold property. Howard Fitch, is planning to direct development of a placer property in southern Utah.

Chief Consolidated is undertaking two development programs at present, according to Cecil Fitch, general manager. One involves sinking the Apex Standard shaft and searching for the continuation of the gold orebodies contained in the adjoining Eureka Standard property, now the largest gold producer in the district. The other program involves drifting from the 1,700 level of the North Lily shaft of International Smelting to the middle of the Eureka Lilly property. This area has not been developed previously. Production from a new lead-silver-zinc deposit found below water level at the Chief mine is being postponed until metal prices improve.

Silver King Adds to Ore. During 1931, Silver King Coalition completed 25,364 ft. of new development work, according to M. J. Daily, general manager of the property at Park City, Utah. Exploration in virgin territory proved continuity of the ore-bearing zone, and reserves far in advance of extraction requirements have been blocked out. As a result of improvements in mill practice, the grade of lead concentrate increased to 67.3 per cent lead and of zinc concentrate to 61.2 per cent zinc. Regular employment was afforded 500 men by operations in 1931, which, W. M. Ferry, managing director, states, were not carried on for profit but rather to meet operating expenses while providing work.

Bisbee, Ariz., Feb. 26.

Shattuck Denn Concreting. Concreting of the Denn shaft, of Shattuck Denn Mining, which was started from the 2,200 level last September, should be finished by the middle of 1932. Unless decided improvement in the copper market is then shown, the company will suspend all work. Drilling of the new orebody east of the Denn shaft and be-

low the 2,200 level indicated an important orebody. The company produced no ore in 1931, either from the Denn or Shattuck mines, the annual report states. Some drifting was done on the 2,100 and 2,200 levels without finding additional ore.

Miami, Ariz., Feb. 27

Longer Runs for Two Arizona Smelters. The Phelps Dodge smelter at Clifton and the International plant at Miami are lengthening their periods of alternate activity and inactivity. New schedules are experimental, but will be followed unless found inexpedient. Clifton shut down Feb. 15 for a month and will probably run for the month following March 15. In January it ran three days each week, fires being banked on the other four. International smelter was idle all of February, but will be operating after March 1. It will be running almost entirely on Miami concentrates.

Inspiration Staggering Work. Inspiration Consolidated Copper, which is mining only ore which can be treated at its leaching plant, is staggering employment so as to give work to as many men as possible. Some men on shift work get in 18 shifts a month, others only 12. Most of the office force are working 21 days per month.

Reno, Nev., Feb. 26

Gold Hill Output Down. The Gold Hill Development, at Round Mountain, in January shipped about $24,000 in gold bullion. The total for February is expected to fall $1,000 or so below this, on account of the short month. The Gold Hill statement for year ended Dec. 31, 1931, shows gross output of $311,919. Disbursements were $296,423, a net of $15,495 for the year. A large amount was expended during the year in new work and installation of heavy-duty pumps. A heavy water flow was encountered on the 500 level that delayed work on this level and called for extra pumps. At last report the 500 level was showing good-grade milling ore. The mill has been kept running all winter despite the unusually cold weather. This was possible because a heating plant was installed last summer.

Tonopah Mill Still Running. Although most Nevada mining camps have been snow blockaded since before Christmas, and but little ore was sent to the custom mills, the Tonopah mill, at Millers, has kept running on ore from Tonopah and Goldfield leases. The output for last half of January was twelve gold-silver bars of bullion valued at about $12,163. It has not been possible to ship the ore stored at the Esmeralde, at Gilbert, to the railroad nor ore in from Ellendale and other camps near Tonopah.

Newpass Pays Dividend. Newpass Mining is the first new Nevada mining company to pay a dividend in 1932. The property is about 25 miles northwest of Austin. The ore from which the dividend was paid was drawn from the Newpass property, which has been opened by two tunnels. A third tunnel is being driven to open the mine at lower depth. The company also owns the Superior mine, on the other side of the gulch from the Newpass. A compressor and other equipment have been installed at the Superior mine, and a tunnel has been started to explore the vein system. An old 400-ft. shaft will be reconditioned and sent to greater depth. Newpass is equipped with a 50-ton mill, about 5 miles down the canyon from the tunnel portal. The ore runs about $13 per ton in gold, and output is about $18,000 per month. Ore from the Superior will go to the same mill.

Walker Mill Closes. The flotation plant at the Walker copper mine, 8 miles from Spring Garden, Calif., is to close March 1. Mine operations will be continued by 100 men, preference being given to married men. All underground bosses will be retained and the company will continue to provide houses, lights, and fuel, and pay a living wage. Until about a year ago, Walker employed about 420 men. The force was then cut to 350 and now to 100. Concentrate is shipped to the International smelter in Utah, as the company is controlled by Anaconda Copper.

Silverton, Colo., Feb. 15.

Shenandoah-Dives Costs Down. By operating at the capacity of its present plant, Shenandoah-Dives Mining, gold-copper producer near Silverton, succeeded in reducing mining and milling costs from $4.22 a ton in 1930 to $3.30 in 1931. Still further reductions were made toward the end of the year. In December, when the mine produced 22,000 tons of ore, mining costs were down to $1.30 per ton, compared with the 1931 average of $1.79. As a result of development work, ore in sight has been considerably increased. Mill capacity, originally rated at 300 tons daily, has been demonstrated to be in excess of 575 tons. Provisions of additional Dorr classifier equipment seems the major requirement for further expansion in capacity.

[Associated Press reports to New York state that several snowslides have blockaded the Silverton-Durango branch railroad and the mining community at Silverton is now isolated from outside contact. It may be several weeks before the blockades can be removed.—*Editor.*]

Deadwood, S. D., Feb. 19.

Black Hills Cons. Formed. Black Hills Consolidated has been formed to take over an area of 8,600 acres in the Lead gold district of South Dakota. The company has an option over 3,000 additional acres. Among the claims obtained are those formerly owned by United Mining, Golden Reward, and North Homestake. O. R. Whittaker, president of Ahumada Lead, heads the new company. The only present substantial gold producer in the area is Homestake Mining.

Annual Reports of Mining Company Earnings

Company	Minerals Produced	Location	Operating Profit	Net Profit	Net Profit Previous Year	Value Output	Dividends Paid
Ahumada Lead	Lead, silver...	Mexico	—$95,890	—$148,126	—$153,240	$132,482	None
American Metal	Various	U. S., Mexico	$1,667,496	—$615,909	$1,827,579	(a)	$307,980
Am. Metal of New Mexico	Zinc, lead	New Mexico	—$437,129	—$1,259,573	—$1,125,494	(a)	None
Barry Hollinger	Gold	Canada	$5,080	(a)	(a)	$224,456	None
Chief Consolidated	Lead, silver..	Utah	—$112,797	—$158,664	—$69,901	$209,405 (b)	None
Coniaurum	Gold	Canada	$191,112 (c)	$22,069 (c)	—$55,109	$785,708 (h)	None
Cusi-Mexicana	Silver, lead...	Mexico	—$63,248	—$153,832	—$14,621	$245,190 (b)	None
Howe Sound	Lead, copper.	Canada, Mexico.	$1,169,964	$593,215	$2,030,292	$8,593,678	$1,100,030
Minas de Matahambre	Copper	Cuba	$124,545	—$321,256	$572,843	(a)	$46,000
Mohawk Mining	Copper	Michigan	—$76,483	—$9 513	$432,126	$764,094	$111,700
National Lead	Lead, tin	U. S., Argentina..	$10,622,213	$4,022,421	$4,675,098	$50,856,343	$3,952,006
Premier Diamonds (d)	Diamonds	South Africa	(a)	—£190,240	£62,500	£278,400 (b)	None
St. Joe Lead	Lead, zinc	Missouri, N. Y...	$1,811,188	—$1,409,353	$1,509,991	(a)	$1,755,419
San Francisco (e)	Lead, zinc	Mexico	£60,895	£19,033	£88,361	(a)	None
Shenandoah-Dives	Gold, copper.	Colorado	$116,177	(a)	$41,375 (f)	$680.406 (b)	None
Sherman Lead	Lead, silver..	Idaho	$4,125	(a)	$75,927 (f)	$27,852	None
Silver King Coalition	Lead, silver..	Utah	$26,849	—$10.137	$544,533	$1.241,908 (b)	$183,070
Texas Gulf Sulphur	Sulphur	Texas	(a)	$8,942.602	$13,972,085	$18.213,807 (g)	$8,255,000
United Verde Extension	Copper	Arizona	$350.029	—$382.160	—$1,513,861	$2,342,832	$1,575,000
Wright-Hargreaves	Gold	Canada	$1,126,136 (h)	$1,455,231 (c)	$1,186,004 (c)	$2,912,309 (h)	$550,000

In general, the figures for operating profit (or loss) represent the balance after deducting cash expenditures from the value of production. The figures for net profit (or loss) include the addition of sundry income, and the deduction of charges for depreciation, taxes, or other extraordinary expenditures. Because of the difference in accounting methods of different countries and different companies, the figures are not strictly comparable.

All items for year ended Dec. 31, 1931, unless otherwise specified.

(*a*) Not available. (*b*) Net proceeds from sales of metal, ore, or concentrate. (*c*) Includes premium on Canadian exchange. (*d*) For year ended Oct. 31, 1931. (*e*) For year ended Sept. 30, 1931. (*f*) Operating profit for previous year, net not being available. (*g*) Total income, including other than that from production. (*h*) Does not include premium on Canadian exchange.

Boise, Idaho, Feb. 24

Ship Fuel by Airplane. A shipment of fuel oil is to be made by airplane to permit continuation of gold milling at Stibnite, Idaho. Heavy snows prevent delivery of additional supplies of oil by truck and the existing supply at the camp has been lost through pipe leakage. About 3,000 gal. will be transported over this route, three planes being used.

Joplin, Mo., Feb. 24

Childress Mill Burns. The 300-ton zinc-lead concentrator of Childress & Heggam Mining, in the Cardin (Okla.) section of the Tri-State district, was destroyed by fire on Feb. 18 with a loss of about $60,000, according to Fred Childress, manager, Joplin. The plant was in operation at the time. No decision is reported to have been made as to rebuilding.

Minerals Beneficiation Formed. Formation of the Minerals Beneficiation, Inc., to promote commercial development of the differential density process of mineral separation, as worked out by C. Erb Wuensch, Joplin, mining engineer, has been announced by Victor Rakowsky. Mr. Wuensch has been working on the mechanism for the last few years and operated the original unit in a mill of Missouri-Kansas Mines, in the Waco district. Recently a unit of approximately 30-ton per hour capacity was installed at the coal plant of Spencer Coal, at Pittsburg, Kan., and the results were so gratifying to C. F. Spencer and Kenneth Spencer, owners, that they have become interested in the development of the machine and have made arrangements to install a larger unit, capable of washing around 2,000 tons of coal a day, at their plant.

The company expects to develop the use of the process in all classes of mining and has made arrangements for its handling and placement in the coal industry with The Roberts & Schaeffer Company, leading coal preparation and plant construction group, Wrigley Building, Chicago.

The process makes use of a cone, which is filled and fed a high specific gravity medium on which the barren rock will float and be discharged and through which the ore-bearing particles will sink. The owners claim that the process will allow for coarse grinding and in addition give off a barren product of coarse rock that is highly desirable in construction work, besides making a more economical separation.

Washington, D. C., Feb. 27

A successful drive against Bureau of Mines appropriations was made in the House on Feb. 22. The cuts, totaling $194,615, are in addition to a reduction of $186,253 which previously had been cut out of the bill by the Appropriations Committee. With the reductions previously made by the Secretary of Commerce and by the Bureau of the Budget, in line with the general movement for the reduction of government expenditures, this would mean that the promotional work of the federal government on behalf of the mining industry will be reduced $420,430 over the amount appropriated for the current fiscal year. This is a reduction of nearly 20 per cent. If the Senate does not restore the amounts cut out on the floor of the House, a drastic change will have to be made in the work of the Bureau of Mines.

Houghton, Mich., March 1

Copper Range Curtails Further. No. 1 shaft, Champion mine, was shut down on March 1, and the men were transferred to Nos. 3 and 4 shafts, now the only operating units. These shafts will be in commission six days a week, but the men will work only three days a week instead of four as formerly. Two boiler houses will be closed. No. 1 shaft will be kept dry and in repair. As a result of these steps production will be further reduced.

Onondaga Will Dissolve. At a special meeting, held in Houghton, stockholders voted to dissolve the Onondaga Copper Company and to pay a partial liquidating dividend of 15c. per share on April 15 to stockholders of record April 1. About $19,000 is in the treasury in cash and Liberty bonds. Authorized capital stock is 150,000 shares, of which 100,000 are outstanding. The company's mineral acreage of 10,000 acres will not be sold at this time. The property was drilled systematically for four years with disappointing results.

New York, March 2

American Smelting Reports. In 1931 American Smelting & Refining made a net profit of $874,976 after interest, depletion, depreciation, and Federal taxes, or $1.75 a share on 500,000 shares of 7 per cent, preferred, compared with 1930 earnings of $3.77 a share on the 1,829,940 shares of common, after dividends on the preferred stock. Current assets at the end of the year were $69,163,087 and current liabilities $10,979,908. Preferred and common dividends totaled $8,359,927, leaving a deficit of $7,484,951. The deficit in the year preceding was $429,346. Fall in metal prices called for inventory adjustments totaling $3,194,987.

Mohawk May Liquidate. Directors of Hohawk Mining, operating near Calumet, Mich., have for several months been giving consideration to the desirability of liquidating the company's affairs, according to L. P. Yandell, president. "Their reasons are chiefly the uncertainty surrounding the present situation of and future outlook for the business of mining copper, together with the short life remaining for the company's mine," Mr. Yandell stated. "Engineers estimate this remaining life to be two years from Jan. 1, 1932, at the present maximum producing rate, followed by one or two years of declining production."

According to the annual report, Mohawk produced refined copper at an average cost of 7.038c. a pound in 1931, and though this was the lowest cost on record for the company, it had been further reduced in 1932. The current producing cost was estimated at under 6½c. a pound. The company mined 448,564 tons of ore last year and produced 18,686,200 lb. of mineral, including 13,100,000 lb. of copper. Current operations are on a six-day basis.

Kennecott and Inco Dividends. Four of the leading non-ferrous metal producers in the Americas—Utah Copper, Nevada Consolidated, Kennecott, and International Nickel—have decided not to pay dividends during the current quarter. The action was generally expected. The first three companies, belonging to the Kennecott group, paid dividends of $1, 10c., and 12.5c. a share last quarter. International Nickel paid 5c.

St. Joe's New Acid Plant. At Josephtown, Pa., the acid plant of St. Joseph Lead, put into operation in 1930, has operated satisfactorily. About 10,400 tons of 100 per cent sulphuric acid was produced in 1931. This is about 30 per cent of capacity. The zinc smelter started in January, 1931. Original estimated furnace capacities and costs are being equaled.

With regard to the zinc mines in New York, the new No. 2 shaft at the Balmat mine has been completed. Development is satisfactory. On the 1,700 level of the Edwards mine, additional tonnage has been found to prolong the property's life. The company also reports that the new 200-ton flotation plant to treat gold ore from the Boise-Rochester mine, near Atlanta, Idaho, has been constructed. Operation will start this year.

U.V.X. Finds No New Ore. James S. Douglas, president of United Verde Extension, operating at Jerome and Clemenceau, Ariz., reports that virtually no new ore was found in 1931 at the mine. As a result, ore reserves decreased by the tonnage mined—or about 183,000 tons—to 440,000 tons, averaging 7 per cent copper. Costs were reduced to 7.14c. a pound. In Haynes ground, where new mineralization has been found, sinking of the Haynes shaft will be deferred for the present because United Verde Copper is driving an exploratory drift into that ground at 1,500 ft. As yet no commercial material has been found in Haynes ground.

Mining Company Dividends for February, 1932

Companies in the United States	Situation	Per Share	Total Disbursements
Alaska Juneau Gold	Alaska	$0.125Q	$180,750
American Smelting & Refining	Various	0.125Q	228,743
Homestake Mining, g	South Dakota	1.65MX	414,414
National Lead, pd., "B"	Various	1.50Q	154,913
New Jersey Zinc	Various	0.50Q	981,632
United Verde Extension, c	Arizona	0.25Q	262,500
Companies in Other Countries			
Cerro de Pasco Copper	Peru	0.25Q	282,710
Granby Consolidated, c	Canada	0.125Q	56,250
Hollinger Consolidated, g	Canada	0.05M	246,000
International Nickel, pfd	Canada	1.75Q	158,959
Teck-Hughes Gold	Canada	0.15Q	713,574
Total			$3,680,445

c. copper; g, gold; Q, quarterly; M, monthly; X, extra.

CANADA

Vancouver, B. C., Feb. 24

Hecla Options Maple Leaf. Hecla Mining has secured an option on the Maple Leaf claim, near Franklin, in the Boundary district. The company holds the Union group of claims on which active operations are in progress. Wilfley tables have been installed in the Union mill and concentrate said to be worth $12,000 per ton is being shipped to the Department of Mines at Ottawa. This ore is being taken from the vein, newly discovered in the Union workings, from which carload shipments of exceptionally high-grade gold ore were made recently. The Maple Leaf claim lies north of the Union claim abutting on to it at one point and was previously held by the company, but the option was dropped. The present purchase consideration is said to be $50,000.

Reno Developing at Depth. The Reno mine, near Salmo, in the Ymir district, operating a 45-ton treatment plant, has been responsible for production of around $400,000 since milling started in 1929. The present annual rate is about $250,000. The steady increase in production is due largely to metallurgical advances, but improvements in the width, average grade, and the lateral extent of the oreshoots have been found to occur in depth. The vein is being developed at a further depth of over 250 ft. below the present workings by means of a long crosscut tunnel on the No. 5 level.

Pioneer Output and Profits. Production in 1931 from the Pioneer mine, Bridge River, was about $609,000. At present, the net yearly profit is estimated at about $400,000, as compared with about $75,000 twelve months ago. When development and mill construction now under way are completed, production is expected to be three times as great. This new stage should be reached late in the year, when the vein is expected to be opened to a further depth of 625 ft. through the new shaft that is being sunk.

A Vancouver syndicate, with which Brig. Gen. J. Duff Stuart is identified, is reported to have acquired extensive placer holdings in the Barkerville area. This ground includes the alluvial deposits of Burns Creek, Lowhee Creek, Stouts Gulch, Mosquito Creek and Conklin Gulch, from which gold worth over $1,000,000 was taken in the twenty-odd years that hydraulic operations were conducted by the original owners. Active operations are expected to start as soon as weather conditions permit.

Oro Fino May Reopen. Arrangements are reported to have been completed for reopening the Oro Fino mine, in the Osoyoos district. A small stamp mill was erected on this property several years ago to treat high-grade gold ore occurring in the form of pockets and stringers in quartz veins. More recently a narrow vein with a spectacular gold content was discovered in the course of development. The government engineer has reported favorably on the possibilities.

Activity in Nelson Gold Mines. Considerable interest is being shown in the gold ores of the Nelson district. The Perrier mine is now shipping concentrate to Trail at the rate of about 150 tons weekly; the Venus, adjoining the Athabasca mine, on Toad Mountain, has been reopened by H. D. Dawson and associates, of Nelson; the Nugget-Motherlode, at Salmo, is stated to be the subject of negotiations with Wisconsin interests; operations are in progress at the Euphrates mine, on the road between Nelson and Ymir; and a prospecting syndicate is being formed to conduct investigation in the Sheep Creek area.

Toronto, Ont., March 1

Radium Refinery Planned. Eldorado Gold Mines, with radium-silver properties at Great Bear Lake, has announced that it is negotiating for the erection of a radium refinery to be built this year. The company's property consists of two principal groups, the original discovery at LaBine Point and the Bonanza group about 6 miles away. A the LaBine Point group, No. 1 and No. 2 veins have been developed by test pitting and trenching and have been opened up for a distance of 1,350 ft. by fourteen test pits, all of which show silver and pitchblende.

The pitchblende occurs in a vein formation, in widths ranging from ½ in. to 18 in. In one open cut the pitchblende averaged 10 in. in width for a length of 35 ft. More than 12 tons of high grade pitchblende has been taken out to date, of which 8 tons have been shipped.

Nos. 1 and 2 veins are believed to be converging, and 3,800 ft. north of No. 14 trench a crosscut 50 ft. long uncovered what is believed to be the northern continuation of these zones. A mile still further to the north, being a total of 2 miles from LaBine Point, a similar zone was found showing four separate veins with pitchblende and silver.

At the Bonanza group, the highest silver values have been found, where silver showings have been traced for a mile in length. Although only a comparatively small amount of work has been done in this area, several veins with rich silver showings have been discovered.

Great Bear Lake Discoveries. Silver discoveries have now been made on eight different properties in the Great Bear Lake country and fourteen different companies will be operating there this year. Freight rates have been reduced to $320 a ton for ingoing freight and $160 per ton for outgoing freight.

Sherritt Gordon Results. Despite low copper prices during 1931, Sherritt Gordon Mines, of northern Manitoba, is understood to have made a small operating profit for the year. Total costs per pound of copper, before write-offs and taxes, were about 6c. per pound; copper content of the ore milled averaged 3½ per cent; extraction was about 95 per cent; and copper concentrates averaged 22 per cent.

Noranda Extends "H" Orebody. On the 2,500-ft. level of Noranda Mines development work has disclosed ore 400 ft. wide that is believed to be the downward extension of the "H" orebody. A length of 500 ft. has been shown by drifting and drilling. Assays are not available for the 2,500 ft., but it is understood that the ore will average between 3 and 4 per cent copper and about $4 in gold. Copper output of the property has been reduced, but gold production is being maintained at the rate of about $5,000,000 a year.

Tailing Re-treatment. Wright-Hargreaves Mines has contracted to join Lake Shore in the construction of a tailing re-treatment plant, estimated to cost $250,000 of which amount Wright-Hargreaves will furnish $80,000. A substantial profit is believed to be possible from the re-treatment of tailing on the two properties.

Teck-Hughes Production. During January the Teck-Hughes mine, in Kirkland Lake, produced a total of $540,000 in gold. This payment in New York funds would be about $630,000 in Canadian funds. Total costs, including taxes and depreciation, are between $240,000 and $250,000 per month. Dividend requirements are $240,000 per month. Development results on the lower levels, the 30, 31 and 32, have been satisfactory. In addition, new ore has been found on the 27, 28 and 29 levels, which had been reported as being much below the average of the mine. New ore has also been found on the 700-ft. level, and the ore position of the property as a whole is stated to be satisfactory.

Dome Mines Development. Dome Mines, of Porcupine, is obtaining favorable results on its drilling of the Jowsey gold properties, in Louvicourt Township, in northwestern Quebec. Work consists of trenching and drilling. The surface outcrop has been exposed for a length of 160 ft. The option price on the properties is understood to be $250,000, of which $5,000 has been paid.

Treadwell Yukon to Erect Mill. Treadwell Yukon, which has an option on the Bussiere-Massicotte property, in Louvicourt Township, will erect a mill on the property. Initial capacity of the mill will be 25 tons a day, and will employ equipment taken from the Errington Mine at Sudbury. This equipment will be taken in as soon as possible, to take advantage of the snow roads and save several months' time. Surface work and diamond drilling have given excellent indications, but the only satisfactory way to sample the property properly is believed to be by a mill run. The company has also filed on a water-power site eight miles from the property, which is capable of developing 1,000 hp.

Sudbury, Ont., Feb. 29

International Nickel Starts Plant. On March 14 the International Nickel Company will commence operation of its new Orford separation plant, recently completed in conjunction with the new 5,000-ton nickel-copper smelter at Sudbury. The plant consists of cupolas and converters for the handling of matte from the nickel-copper smelter. Matte containing about 42 per cent copper, 37 per cent nickel, and 20 per cent sulphur is smelted with nitre cake, acid sodium sulphate, to produce copper "tops" and nickel "bottoms," which, in turn are smelted further and shipped in the form of blister copper and nickel sulphides. Blister copper goes to the Ontario Refining Company's plant, less than a mile away, and the nickel sulphides are sent to Port Colborne, for final refining.

By moving the Orford separation process to Copper Cliff another step has been taken in the concentration of International Nickel Company's metallurgical operations

in closer proximity with the ore supply. Moreover a decided economy will be effected in transportation costs. Whereas formerly the entire product of the smelter had to be shipped to Port Colborne for separation, and the blister shipped back to Copper Cliff for refining, under the new arrangement only the nickel sulphides, or less than half the volume of the matte, will be shipped outside the Sudbury district for refining. The freight haul from Copper Cliff to Port Colborne on shipments of nitre cake for separation will also be saved by the new arrangement described.

Three Mills to Be Built. Three new mills for the production of gold will be erected this year in northwestern Quebec. The largest will be that of the Beattie Gold Mines, Ltd., in Duparquet Township, which will have an initial capacity of about 1,500 tons and treat ore having an average grade of about $3 per ton. In Cadillac Township, O'Brien mining interests are reported to be preparing to erect a mill of about 100 tons' daily capacity. Some high-grade ore has already been bagged and shipped direct from the property to the Deloro refinery. The third mill is a testing plant that Treadwell Yukon Mining will install on its Louvicourt Township claims. Ball mills and classifiers will probably be moved from the mill on the Errington mine at Chelmsford, near Sudbury, to meet any requirements that may arise from the company's operations in Quebec.

Grozelle Kirkland Control Changes. With increasing development at depth of Kirkland Lake mines, some attention has been given to the possibility of values being found on the dip of the veins into adjoining properties. Kirkland Lake Gold acquired adjoining claims of the Chaput Hughes Gold Mines, Ltd., two years ago. Recently it has obtained control of the Grozelle Kirkland Gold Mines, Ltd., with claims adjoining it on the north. The latter has an authorized capital of $3,000,000 in $1 shares, of which 2,182,858 shares are issued.

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PANAMA

London, England, Feb. 23

Panama Corporation Reconstruction Proposed. A scheme has been proposed whereby all assets are to be transferred to a Canadian company with a capital of 4,500,000 shares of no par value. The new company will create $5,000,000 8 per cent convertible debenture stock. Present shareholders will receive one share of no-par value for every share held, and a holder of £100 of existing debenture stock will receive $500 of similar stock in the new company, conversion rights remaining as at present. Operations on the Darien area have been stopped owing to losses incurred in remitting working capital to Panama. Remance mill is working at capacity. Diamond drilling is in progress at Mina Blanca and on the Chiriqui line. Two lodes have been opened at Capira on the Rodriguez concession. The January return from Remance was 4,004 tons averaging 25.5s. The mill treated up to 142 tons per day, the tailing averaging 2.87s.

MEXICO

New York, N. Y., Feb. 26

Potosi Reserves Higher. Development in the Potosi Mining lead-zinc property, near Santa Eulalia, Chihuahua, added materially to ore reserves in 1931, according to the annual report of Howe Sound, which controls the property. Some lead and zinc concentrates are being stored pending a better market. The mine, the largest lead producer in Mexico, has curtailed output somewhat.

Kildun Developments. About 300 men are now engaged in development work at the Kildun silver-gold property, at Matehuala, S.L.P. Installation of additional power equipment has permitted pumping out water on the bottom or 445-m. level and development there has been resumed. On the 300-m. level, the gold content is low, but the main vein shows ore averaging more than 32 oz. in silver to the ton over widths of 8 to 10 ft. About three carloads of gold ore are shipped daily. As work in the adjoining Santa Maria de La Paz property has indicated continuation of the Kildun vein to 560-m. level, the management expects to find ore at least 400 ft. below its present deepest level.

Ahumada May Sell Mine. Ahumada Lead, owner of a lead-silver mine in the Los Lamentos district, Chihuahua, is looking for a purchaser or lessee of the property, which was closed down on June 30, 1931. Efforts have to date met with no success and the company is seeking to dispose of its equipment in smaller units. Low metal prices and excessive metal flow have prevented exploitation of ore deposits below water level.

London, England, Feb. 23

San Francisco Mines Position. Ore reserves of the San Francisco Mines of Mexico, in Chihuahua, are 1,837,270 tons blocked out and 483,670 tons partly blocked out, an increase of about 50,000 tons, the chairman announced at the recent meeting. Only sufficient ore has been developed to replace that sent to the mill. Plant and workings are being kept in order, but no profit can be made at present metal prices. Working costs had been reduced to 14s. 6d. per ton, against 19s. 1d. in 1928-9. Possibilities of further improvement in metallurgical practice are limited. Since operations were suspended, base-metal prices as quoted in sterling have risen 35 or 40 per cent, but as the bulk of the company's expenses are payable in dollars, the revenue must be calculated at its dollar value and the metal prices at their gold values, which have hardly risen since September last.

Fresnillo Development. Because of the fact that development work on the sulphide orebodies at the Fresnillo property, Fresnillo, Zacatecas, is well in advance of mill requirements, it is being somewhat curtailed. The company's most recent quarterly report, however, shows that drifting on the three new levels—at 470, 515, and 560 m.—continues to open new ore. Drifting west on the east oreshoot on the 570-m. level had opened 853 ft. of ore up to Dec. 31. This ore was 4 ft. wide and averaged 7.8 per cent lead, 10 per cent zinc, 2 per cent copper, 13.5 oz. of silver and 0.2 dwt. in gold per ton. The copper content is considerably higher, and the lead and zinc contents are substantially lower than in the upper levels. Production in the last quarter of 1931 totaled 243,662 tons—59,004 tons of sulphide, 131,113 tons of oxide, and 53,545 tons of tailing.

Teziutlan Closed. No development work was done at the Teziutlan zinc-copper mine, in Puebla, during the fourth quarter of 1931. The mine, which produced 15,035 tons of ore during the period, has now been closed. The quarter showed an operating loss of $56,988.

Spokane, Wash., Feb. 15

Mexican Corporation Options Mine. Mexican Premier, controlled by local interests, has optioned its Elissa gold mine, in Nayarit, near the Sinaloa line, to Mexican Corporation, operators of the Fresnillo and Teziutlan mines. The total purchase price is said to be $200,000 for Mexican Premier's 51 per cent interest and option over the remaining 49 per cent. The Elissa mine is close to the Southern Pacific Railroad. Mexican Premier's silver-lead mines, near Mazatlan, have been closed down.

Duluth, Minn., Feb. 9

Cusi-Mex. Adds Ground. Cusi-Mexicana, operating a silver property in Chihuahua that was closed on Feb. 1, added new ground to its holdings at Cusihuiriachic during 1931, the annual report, issued here, reveals. Among these was the Abundancia mine, adjoining the San Miguel on the east. Although underground development was curtailed in 1931, several new orebodies were opened in the Santa Marina mine. To the south, ore 12 to 15 ft. wide has been opened in the Santa Eduwiges vein. Development of the San Pedro claim gives promise of a large tonnage of cheaply mined ore averaging 25 oz. in silver per ton, according to H. C. Dudley, president of the company.

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ASIA

London, England, Feb. 23

Merger Proposed in Kolar Field. Negotiations for fusing the Balaghat and Nundydroog undertakings have been officially proposed. The former has had difficulty in meeting the heavy cost of deep development and maintaining an adequate ore supply for the mill, and has limited resources for exploration. Nundydroog has been very successful since reorganization in 1920, mainly owing to opening a new rich orebody in the Orient section, which adjoins the Balaghat. The mines in this district are benefiting from the high price obtainable for gold. Champion Reef has announced a distribution of 1s. 6d. per share, the highest rate ever paid by the company.

Kolar Gold Field Results in 1931

	Ore Milled	Ounces	Development Footage
Mysore.....	182,731	96,042	11 892
Champion Reef..	98,930	65,719	8,947
Ooregum.....	135,095	63,023	5,362
Nundydroog....	123,039	79,836	9,077
Balaghat.....	41,850	25,835	4,750
Totals.....	581,645	330,455	40,028

AUSTRALIA

Melbourne, Victoria, Jan. 15

Gold Yield by States. Activity in Australian gold mining during 1931 is shown by increased yields from the principal producing states. In West Australia, production was 510,572 oz., compared with 416,369 oz. and 377,176 oz., in 1930 and 1929 respectively. The 1931 output is the highest since 1922, when the yield was 538,246 oz. The increase is mainly due to starting operations at Wiluna, and the increased tonnage treated at Lake View & Star, Kalgoorlie. In Victoria the yield for the year was 42,500 oz.; in 1930, 24,119 oz. The total number of mining companies, mainly gold, formed in Victoria during 1931 was 36, an increase of 8. In Queensland the yield was 10,321 oz.; in 1930 it was 7,821 oz.

Bendigo's Prospects. In 1932 the gold output from Bendigo is expected to be considerably greater than in 1931, when the yield was 16,013 oz., an increase of 4,373 oz. over the 1930 figures. The principal producers in 1931 were Ironbark, 3,706 oz.; Hercules, 2,866 oz.; and New Red, White & Blue, 2,417 oz. Ironbark paid £6,000 in dividends, equal to 2s. per share. At the New Red, White & Blue, state governmental assistance has made possible the installation of modern electric pump to prospect the lower levels.

About 1,200 prospectors were at work in the district throughout the year. Their contribution was 5,940 oz. or less than 5 oz. per man. Further to assist prospectors, the state government now proposes to crush, at the state batteries, lode material assaying less than 6 dwt. per ton for the low figure of 2s. per ton. Should the material assay more than 6 dwt. the charge will be 6s. The government will also assist in defraying the cost of transporting material to the battery site.

The Victorian Gold Association, formed to unwater and develop the Monument Hill, Deborah, and Nell Gwynne mines, has started operations. A new Bendigo company, Central Blue Gold Mines, Ltd., has been formed with a capital of £30,000 in 60,000 10s. shares, to unwater and further develop the Central Blue mine, which has been worked to a depth of 2,300 ft. Most of the eastern ground remains untouched. Maude & Yellow Girl Gold has been registered with a capital of £60,000 to acquire and work the Glen Wills mine, Glen Wills, Victoria. E. C. Dyason, formerly general manager of the Bendigo Consolidated Mines, is one of the directors.

Broken Hill P. at Kingston. Excellent progress is being made by Broken Hill Proprietary in sinking a shaft on the Kingston lease, at Bolangum, near Stawell. The double-compartment shaft has reached a depth of 210 ft. and will be continued to 300 ft., when the reef is expected to be met. Driving along the strike of the reef will then be undertaken. In addition, the wide Wonga reef is being trenched, and should assays be sufficiently attractive, the outcrop will be mined by open-cutting.

Broken Hill Proprietary has exercised its option of purchase over the Junction Reef mine, at Mandurama, N.S.W. Recent crushings from several flat-dipping lodes have assayed between 5 and 15 dwt. in gold per ton. At present 240,000 tons of ore is available for extraction.

Gold Bounty May End. The suspension of the Federal Government's bounty of 10s. per ounce on new gold is under consideration. The view is taken that as gold is now worth £7 11s. per ounce in Australian currency, a bounty is not justified. On the increased production for the fiscal year 1930-1931, the bounty would total £80,000.

Electrolytic Zinc Curtails. Electrolytic Zinc has decided to restrict the output of zinc from its works at Risdon, Tasmania. Part of the plant was idle during January and 40 employees will be released. No further details have yet been made public. Plant capacity is 4,500 tons of zinc and 20 tons of cadmium monthly. The extensions to increase capacity by 1,400 tons monthly were suspended about twelve months ago.

Charters Towers Tailing. To recover gold from old tailing dumps in the Charters Towers district, Queensland, Charters Towers Gold Recoveries has been formed with a capital of £30,000. In all 3,000,000 tons assaying 8s. 6d. per ton, with gold at 85s. per ounce, is available for treatment. Much of the equipment required is in the district. An extraction of more than 1 dwt. per ton is expected with costs about 25s. per ton. The material is soft and friable, so that fine-grinding costs should be low. With a recovery of 1 dwt. the total gold to be won is 150,000 oz.

North Broken Hill Reconstruction. Modernization of surface plant and erection of a new mill at the North Broken Hill property, in New South Wales, is making rapid progress. A new 120-ft. all-steel headframe is being built and Metropolitan-Vickers electric hoists are being installed. Automatic tipping arrangement will form a feature of the headframe. Two ore trucks will be used per cage when it is necessary to supplement ore haulage in skips. An ore bin of 100 tons' capacity is built into the headframe. It feeds into the crushing plant, consisting of a 36x24-in. jaw crusher and 5½-ft. Symons cone crushers reducing to ⅝ in. Crushed ore is delivered to the mill bins by conveyor over a distance of 1,000 ft. The two new hoists were to be operated by direct-coupled, 1,000-hp. motors, and will have 10-ft. driving drums. The skip hoist will be able to handle 200 tons an hour and the cage hoist can, if required, handle an additional 100 tons hourly.

The new mill will be operated in four sections, ore being ground in four 36x18-ft. roll crushers to ½ in. before reaching the ball mill unit, consisting of four 4x8-ft. Ruwolt units. The product, a minus 40 mesh, will be classified in Dorr machines and, after thickening, will be concentrated in a type of flotation machine specially developed in the North mill. Concentrate will be dewatered in Dorr filters. Wilfley pumps will handle the pulp. The mill will be in operation by June and the new shaft equipment by July.

Mount Isa Changes. To increase the output at Mount Isa, Queensland, three new Dwight-Lloyd blast roasting machines are being installed and two additional blast furnaces. The Cottrell dust precipitation plant is being enlarged to take care of the extra fume. As would be expected in starting a lead smelter in a hot, dry, dusty locality, Mount Isa has experienced lead poisoning among its employees. For the period July 1 to Oct. 17, 1931, 68 cases were reported, of which seventeen were recurrent. None was serious. The treatment of lead carbonate ore, the dust from which is more easily assimilated in the system, has no doubt intensified the hazard. The management hopes that as soon as operations are normal there will be no difficulty in stamping out the trouble, as has been so effectively done at Port Pirie, South Australia.

London, England, Feb. 23

Wiluna's New Lode. A 450-ft. level crosscut has cut a new lode about 85 ft. west of the present West lode. The new ore averages 74s. over 2 ft., and drifting has begun. At 130 ft. south, borehole results were 52s. over 17 ft. From the 625-ft. level a borehole at 328 ft. north cut the lode, averaging 61s. over 13.5 ft. Total length thus far exposed appears to be 458 ft.

Kalgoorlie, W. A., Jan. 21

Great Boulder Proprietary Active. The company is pursuing an active development policy and endeavoring to increase tonnage to mill capacity. The downward continuation of the "X" lode, worked on tribute by Boulder Perseverance between the 900 and 1,100 ft. levels, and known to be patchy, with telluride interspersed with low-grade ore, is being worked by the Proprietary company at the 1,650-ft. and 1,800-ft. levels. It has been found to be somewhat similar to the lode as developed by the Perseverance, although no telluride patches have yet been found. Great Boulder has permission from Lake View and Star to open up orebodies at the north end of the Ivanhoe which extend into Great Boulder ground. To facilitate the work, the company is installing a new Bellis-Morcom twenty-drill compressor. Arrangements have been made to do diamond drilling work.

North Kalgurli Rehabilitation Proceeding. Steady progress is being maintained on the surface and underground with the rehabilitation work on the North Kalgurli mine. Development is proceeding. On the surface, the new hoist at the main shaft is in operation. The old one is being re-erected on the Union Jack shaft.

Donnovanville Tin Promising. Developments at Donnovanville between Nannup and Greenbushes, in the southwestern part of Western Australia, continue promising. The new shaft is down 20 ft. Hand borings have been put down a further 30 ft. through a zone of free-sluicing dirt (mica schist), 3 ft. 6 in. wide, not yet exposed in the footwall, and running ¾ lb. to the cubic yard and with tin values of 70 per cent. Several parcels have been treated.

AFRICA

Johannesburg, Transvaal, Jan. 19

Randfontein's New Shaft. Randfontein's new No. 2 North Shaft came into commission a few days ago, and is now supplying substantial tonnages of ore to the mill, on the West Rand, Transvaal. With the completion of this shaft, the company is no longer dependent upon the South and North vertical shafts for the maintenance of the huge tonnage requirements of the mine. In the past the mine has had to operate its two main shafts at full capacity. A breakdown in either would have involved serious consequences, not only as regards tonnage, but also the carrying on of the extensive development program that has been under way in recent years. Mill tonnages have been constantly increasing and are now about 250,000 tons monthly. Completion of this shaft denotes the beginning of a new era at Randfontein, which is the fourth largest gold producer in the world. Drifting south of Nos. 29 and 30 rock tunnels from No. 7 shaft to connect with No. 2 North and North Vertical shafts is proceeding. Once completed, this work will make a large area of new ground available.

West Rand Starts Shaft. West Rand Consolidated started sinking its new South shaft on Dec. 31 and has since maintained a steady pace. The shaft is 1,500 ft. east of the Flora Battery Reef shaft, on the West Rand, Transvaal, and is connected therewith by means of an electric tramline. The new shaft will be 4,200 ft. deep and will hoist the entire ore output from the Main Reef series. The East and West shafts, which have been handling Main Reef ore, will be used solely for ventilation in the future. The company hopes to complete the new shaft in fifteen months.

Development of the Main Reef series through the South shaft will be carried on in the Nos. 27 to 40 levels. A haulageway will be driven in the footwall of the reef on the No. 32 level, extending the entire length of the property. Additional haulageways will be opened up at four-level intervals. Below the No. 40 level, development will be through two sub-incline shafts. The South shaft will be connected to the Battery Reef series workings by a crosscut.

London, England, Feb. 23

Transvaal January Output. Production of gold by mines in the Transvaal in January totaled 936,784 oz., more than has been produced in any single month except October, 1931, when 945,113 oz. was recovered. Of the January output, 890,668 oz. was from Witwatersrand mines. In January, 1931, Transvaal output was 914,576 oz. The principal factor in the auspicious start of the new year was the increased supply of colored labor, 215,572 "boys" being employed, compared with 209,442 "boys" the year before.

Daggafontein's First Month. Daggafontein Mines, new East Rand producer, which started operations in January, treated a total of 21,000 tons. Absorption of gold in the plant kept the recovery down to £13,556. Capacity is 50,000 tons. The new ore reserve estimate for this property shows a considerable increase over the tonnage at the end of 1930. (See table of new Rand ore estimates for details.)

Roan Antelope's Report. The report for the fourth quarter covers the last three months of normal initial operations, curtailment beginning when the period ended. During the quarter the concentrator treated 532,800 dry tons, averaging 3.61 per cent copper (0.48 per cent in oxides). Ratio of concentration was 20 to 1; concentrate averaged 55.57 per cent copper. Of total copper 75.25 per cent was recovered, or of sulphide 85.07 per cent. Of the concentrate 8,453.34 short tons was shipped to Chrome, N. J., the remainder being smelted at the property, yielding 7,872.02 short tons of 99.68 per cent blister. Total copper in concentrate shipped was 9,478,428 lb., and in blister 15,694,347 lb. Allowing for all charges and losses, including that on exchange (but before interest and depreciation), the total cost of the output as electrolytic copper at New York would have been £31.98 per long ton, or almost exactly 5c. per pound. With exchange at $3.46, costs are: delivering underground ore to coarse crushers, 0.72141c. (4.17s.); milling, exclusive of drying and sacking concentrate (a cost now eliminated), 22c. (1.21s); smelting, 0.49824c. per lb. (£2.88 per short ton). In the cost of mining is included all development work done during the quarter (20,619 ft.), but the work on the Beatty Shaft pump station, sump, and pockets is excluded. By Jan. 1 the construction department had been converted to a mechanical and electrical service department for other operations, only a skeleton engineering and construction crew remaining for completing a few minor items. Number of Europeans on payroll was reduced from 836 to 562, and of natives from 4,597 to 3,805.

Rhokana Smelter to Start. The smelter at the N'Kana plant of Rhokana Corporation will be blown in during March. As the concentrator started operating on Dec. 11, the smelter will have a three months' stock of concentrate to treat.

Rand Ore Reserves

The following figures show ore reserves of leading Witwatersrand (Transvaal) gold mines on Dec. 31, 1931:

	Quantity, Tons	Value, Dwt.
Johannesburg Consolidated Group		
Government mines	10,510,000	8.9
Langlaagte Estate	1,190,000	7.2
New State Areas	2,737,000	8.9
Randfontein	6,324,000	6.2
Van Ryn Deep	2,669,000	6.7
Witwatersrand (Knights)	265,000	5.5
Central Mining Group		
City Deep	1,323,800	6.08
Crown Mines	10,578,940	6.53
East Rand property	3,519,480	6.3
Geldenhuis Deep	448,200	5.48
Modder B	1,034,710	6.55
Rose Deep	762,700	4.5
Gold Fields Group		
Robinson Deep	2,592,000	5.9
Simmer and Jack	1,558,700	6.0
Sub Nigel	1,261,000	17.3
Anglo-American Group		
Brakpan	2,520,890	7.78
Daggafontein	2,003,174	8.09
Springs	3,452,983	9.14
West Springs	2,737,020	5.85

Luiri Gold Areas Output. In January Luiri Gold Areas treated 1,330 short tons of ore and recovered 923 oz. of fine gold. The new reef recently encountered in the company's Dunrobin property averages, as indicated by limited initial development work, about 8 to 10 dwt. of gold per ton over a width of 15 to 20 ft., but further work is necessary before a positive statement of the grade and width can be made.

Rhodesia Broken Hill Production. In January Rhodesia Broken Hill Development produced 35 long tons of 90 per cent vanadium pentoxide and 100 tons of 16 per cent vanadium concentrate.

New York, N. Y., Feb. 19

Refining of Rhodesian Copper. American Metal has closed contracts for a term of years with Roan Antelope, Mufulira, and Rhokana for the refining of such portions of their respective productions of blister copper as these companies desire to have electrolytically refined, the American Metal annual report states. Most of the copper from these new Rhodesian mines will probably not be refined, as its precious-metal content is small. About two years ago, construction of a large electrolytic plant at Liverpool to treat Rhodesian blister was discussed, but no progress has since been made. American Metal operates one copper refinery at Carteret, N. J., and has a large interest in another at Copper Cliff, Ont. The Carteret plant, which can be reached by direct-water shipment from Africa, has a large excess capacity at present, as the International Nickel blister it formerly treated now goes to Copper Cliff.

Diamond Mines Close. The De Beers Consolidated and Premier diamond mines, which together produce the bulk of South Africa's diamond output from pipes, have announced that they will close on March 31. The Premier mine is in the Pretoria district, Transvaal, and the De Beers properties are in Griqualand West, Cape Colony. De Beers owns a controlling interest in Premier and is the largest single factor in the diamond trade.

Brussels, Belgium, Feb. 18

Officials Oppose Closing U.M. In connection with cables from New York stating that American copper producers are suggesting a general closing of all copper mines, Paul Crokaert, Belgian Colonial Minister, told the *E. & M. J.* correspondent yesterday that the government would not allow complete suspension. Katanga, and, to a smaller degree, the entire Congo, depend upon Union Minière for their economic life. Inasmuch as Union Minière is still making profits, he declared, he saw no reason for further curtailment on its part.

Production of Union Minière in 1931 is now unofficially estimated at about 120,000 metric tons, compared with 138,949 tons in 1930. Cobalt and radium output were regulated to meet the demand. Ore reserves of the company are reported to have been doubled by the discovery of new deposits on the Katanga railroad. Working of these new deposits, officials state, will be possible under "particularly advantageous conditions."

Congo Gold Output. Production figures for six Congo gold producers in 1931 total 7,384 k., compared with 6,148 k. produced by the same companies in 1930. Minière de Kindu, which started production only in December, is now producing 9 k. monthly. Kilo-Moto reports an output of 486 k. in January. Congo legislation that makes imperative the construction of durable housing for colored mine employees will be repealed, M. Crokaert states. This will permit gold mines to work low-grade alluvial deposits without building permanent camps. Kilo-Moto has spent 60,000,000 francs in building brick houses for colored workers. Many had to be abandoned when particular areas were exhausted.

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EUROPE

Brussels, Belgium, Feb. 18

Huelva Exports. Exports of pyrites through the port of Huelva, Spain, totaled 1,894,555 metric tons in 1931, compared with 2,563,598 tons in 1930. The entire production of the Rio Tinto, Tharsis, and Pyrites de Huelva mines is exported through Huelva. Rio Tinto shipped 1,024,359 tons; Tharsis, 597,313 tons; and Huelva, 179,726 tons. Other products exported by Rio Tinto included 8,438 tons of blister copper, 8,827 tons of cement copper, 50,200 tons of iron, and 146 tons of sulphur. Tharsis exported 122 tons of cement copper and 3,500 tons of iron. Several shafts operated by Pyrites de Huelva have recently been closed, and further curtailment is considered probable because of depressed market conditions.

Penarroya in Turkey. Penarroya Mining, one of the large French-controlled European lead producers, has formed a company to operate the Balia-Karaidin mines, in Turkey. It has a capital of 1,000,000 Turkish pounds.

London, England, Feb. 23

Trepca Operates Steadily. Trepca Mines, in Yugoslavia, treated 27,257 metric tons of ore in January, producing 3,915 tons of concentrate averaging 75.9 per cent lead and 27.7 oz. of silver per ton, and 3,776 tons of concentrate assaying 49.6 per cent zinc.

During the quarter ended Dec. 31, Trepca's mill was operated 97.77 per cent of the possible total time. The 79,864 metric tons of ore treated averaged 10.70 per cent lead, 8.24 per cent zinc, and 4.57 oz. of silver per metric ton. Indicated recovery was 96.26 per cent of the lead in the ore and 88.92 per cent of the zinc. Working costs, including London expenditure, came to £1 1s. per ton, and, as revenue averaged £s 15s. 5d., profit for the quarter was £57,332. To this was added £5,484 from adjustment of concentrate in stock at the end of the September quarter, making the total net surplus for the December quarter £62,816.

Good progress has been made with the new power plant and the extension of the mill, which will increase the present milling capacity 50 per cent. Concrete foundations for the power-house extension, new boiler, and new 2,250 kw. turbo-alternator have been completed. Capital expenditure for the quarter was £10,130.

The main shaft has been stripped and timbered to a point below the 730-meter level. This shaft connects with the bottom adit at the 610-meter level. Development on the 730-meter level has included crosscutting at intervals of 20 meters; 196 meters of this work showed ore of an average grade of 6.5 per cent lead and 10.4 per cent zinc; 101 meters averaged 8.1 per cent lead and 11 per cent zinc. Some high-grade lead ore was being exposed on this level when the quarter ended. Other development work—the total done during the quarter was 888.3 meters—was of a secondary nature, to facilitate stoping and to investigate ore extension from stopes.

Almagrera, Spain, Jan. 20.

Almagrera Mill Started. Results obtained from starting operations at the new 50-ton flotation pilot plant of Consorcio de Almagrera, S. A., are very satisfactory to date, M. Gratacap, superintendent of the plant, reports. The plant was started on Dec. 31. It produces a concentrate assaying 60 per cent lead and 120 oz. of silver to the ton, said to represent a recovery of better than 95 per cent of the metallic content. Ore is crushed in a jaw crusher and two rolls and then ground to 100 mesh in a 5 x 2½-ft. conical ball mill. American flotation equipment and very salt water are used in concentrating.

Moscow, U.S.S.R., Jan. 25

Soviet Graphite Output. Operation of a new graphite plant at Kishtim, near Sverdlovsk, in the Urals, U.S.S.R., has been started, according to the U.S.S.R. Chamber of Commerce. Production in 1932 is expected to be 670 metric tons of graphite. Up to the present, Russia has been obliged to import its graphite supplies.

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ENGLAND

London, Feb. 23, 1932

East Pool Adds to Reserves. At the 1,515-ft. level, in January, the west drive on the south of the Rogers lode, at the East Pool mine, Cornwall, was advanced from 19 to 54 ft., averaging 32 lb. black tin per ton over 7 ft. At the 1,600-ft. level, the west drive on the north branch of the lode was advanced 5 ft. to 760 ft., averaging 81 lb.

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Production Statistics

Recently published annual reports contain the following data on metal production of leading mining companies:

	1930	1931
Ahumada Lead		**Mexico**
Ore shipped, short tons	23,257	8,224
Lead, tons	4,744	1,436
Silver, oz.	85,076	29,630
American Metal (*b*)		**U. S., Mexico**
Gold, oz.	485,176	522,902
Silver, oz.	28,039,429	30,833,667
Copper, short tons	235,666	168,684
Lead, tons	113,949	84,680
Zinc, tons	31,492	18,301
Acid, tons	75,264	51,203
Barry-Hollinger		**Canada**
Ore milled, short tons	31,725	31,958
Gold, oz.	10,516	10,859
Belliere, Mines de la		**France**
Gold, oz.	7,812	9,741
Chief Consolidated		**Utah**
Ore shipped, short tons	80,319	32,456
Gold, oz.	16,702	5,601
Silver, oz.	998,655	513,299
Lead, tons	3,496	1,362
Zinc, tons	479	426
Coniaurum Mines		**Canada**
Ore milled, short tons	122,972	130,585
Gold bullion, value (*c*)	$736,727	$785,708
Cusi Mexicana		**Mexico**
Ore mined, metric tons	107,374	44,190
Concentrate, tons	7,396	3,254
Silver content, oz.	1,556,188	958,075
Gold content, oz.	(*a*)	1,355
Lead content, tons	2,571	647
Copper content, tons	135	47
Domaniales de Potasse		**France**
Ore mined, metric tons	2,133,000	1,500,000
Potassium chloride, tons	388,000	279,000
Equatoriale de Mines		**French Congo**
Gold, oz.	2,829	8,070
Howe Sound		**Canada, Mexico**
Copper, tons	22,633	14,309
Lead, tons	45,215	48,981
Zinc, tons	29,560	37,198
Gold, oz.	12,770	5,476
Silver, oz.	3,606,977	3,648,279
Kampong Lanjut		**Malay**
Gravel treated, cu.yd.	2,667,000	4,247,000
Tin concentrate, long tons	613	847
Laurium, Cie. Francaise		**Greece**
Lead, metric tons	7,329	6,700
Mohawk Mining		**Michigan**
Ore milled, short tons	472,623	443,940
Copper, tons	6,653	6,550
Mount Elliott		**Australia**
Ore, long tons	(*a*)	6,248
Copper, per cent	(*a*)	23
Premier Diamond (*d*)		**South Africa**
Loads washed	4,298,130	3,275,807
Diamond, carats	700,943	604,217
St. Joseph Lead		**Missouri, New York**
Ore mined, short tons	5,999,813	4,465,794
Lead concentrate, tons	273,614	196,481
Lead content, tons	177,935	141,999
Zinc concentrate, tons	86,795	63,348
Zinc content, tons	50,064	37,056
San Francisco Mines		**Mexico**
Ore milled, metric tons	337,480	393,390
Lead concentrate, tons	43,771	45,727
Zinc concentrate, tons	35,960	47,088
Shenandoah-Dives		**Colorado**
Ore milled, short tons	100,141	170,795
Gold in concentrate, oz.	17,753	30,561
Silver in concentrate, oz.	287,294	408,756
Copper in concentrate, tons	629	740
Lead in concentrate, tons	319	501
Sherman Lead		**Idaho**
Ore mined, short tons	29,847	7,603
Lead, tons	3,154	935
Silver, oz.	210,848	48,448
Silver King Coalition		**Utah**
Ore mined, short tons	155,950	112,778
Lead, tons	18,359	13,058
Zinc, tons	8,511	6,146
Copper, tons	631	397
Gold, oz.	3,112	2,203
Silver, oz.	2,250,871	1,531,236
Sungei Way (*e*)		**Malay**
Gravel treated, cu.yd.	.....	(*a*) 3,796,245
Tin concentrate, tons	977	1,119
Tongkah Harbor		**Malay**
Gravel treated, cu.yd.	.....	(*a*) 2,712,000
Tin concentrate, tons	875	689
United Verde Extension		**Arizona**
Ore mined, short tons	301,562	183,151
Copper, tons	22,784	12,295
Wright-Hargreaves		**Canada**
Ore milled, short tons	220,430	266,352
Gold bullion, value (*c*)	$2,431,896	$2,912,308

All figures for year ended Dec. 31, 1931, and Dec. 31, 1930, except where otherwise specified.

(*a*) Not available. (*b*) Includes ore treated for account of others. (*c*) Does not include premium on Canadian exchange. (*d*) For year ended Oct. 31. (*e*) For year ended June 30.

Weakness in Metals Continues as Trade Lags—New Lows in Copper and Zinc

Demand for non-ferrous metals during February held to the same low level as in the preceding month, and further weakness in prices developed in copper, lead, and zinc. Toward the close of the month the entire market suffered because of the unfavorable nature of the reports on the latest deliberations of the copper producers in reference to further curtailment in production. New all-time lows were established in both copper and zinc. Lead was under pressure on the gain in stocks. Great Britain's general import tariff of 10 per cent was ratified during the month. Copper remained on the British free list, much to the relief of producers in the international cartel. Lead and zinc, however, are dutiable, effective March 1, resulting in the virtual suspension of the trade agreements abroad on these metals. The *E.&M.J.* index number of non-ferrous metal prices for February was 51.30, against 54.87 a month previous.

Copper at 5¾c., Delivered

Owing to the steady accumulation in supplies, even though world production has been reduced to 26½ per cent of rated capacity, copper producers are faced with the problem of further restricting output. Opinion has been sharply divided as to the method to pursue in arriving at the desired result. The meetings held in New York during the last week of the month brought out no solution of the problem. In fact, on Feb. 29 the outlook was so gloomy that many in the industry had given up all hope of a satisfactory settlement. No agreement was reached during February on the revised rules of the export organization, and producers temporarily went back to the old rules, but with certain modifications in respect to the quotas on electrolytic copper. The possibility of an eleventh-hour settlement loomed just as the month came to a close, for representatives of Katanga announced that they were willing to meet the terms of American producers calling for another drastic cut in output.

The drive for further curtailment in production called for reduced operations by all concerned, without special privileges, to a point estimated at about 17½ per cent of rated capacity. This would bring world output to between 62,000 and 65,000 tons. World consumption apparently has declined to less than 85,000 tons a month. World stocks of raw copper above ground in the hands of producers and fabricators are enormous, some private estimates bringing the total between 950,000 and 1,000,000 tons.

Domestic business on the last day of the month was booked at 5¾c. per pound, delivered Connecticut, a new all-time low on transactions made by first hands.

Export sales for February totaled 25,960 long tons, against 16,354 long tons in January, and 25,450 long tons in December. Copper for export sold down to 5⅞c., c.i.f. basis, under the special privilege clause of the export association. This price also constitutes a new low record.

New Low for Zinc

A fair volume of business was put through in zinc, but at further price concessions. Prime Western zinc for shipment over the next two months sold during February at 2.80c. per pound, St. Louis, a new all-time low. Consumption remains at a low level and even at the present greatly restricted rate of production the industry is faced with the danger of an increase in stocks. Stocks of zinc in the United States have held around 130,000 tons for several months now.

Production statistics for November, December, and January, unallocated according to the origin of the ore, except for the United States and Mexico, in short tons, follow:

World Zinc Production

	Nov., 1931	Dec., 1931	Jan., 1932
United States	20,526	21,965	22,516
Mexico	2,972	3,105	2,938
Canada	7,835	7,262	7,478
Belgium (*a*)	12,190	11,883	11,254
France	4,967	4,650	4,785
Germany	(*a*) 4,302	(*a*) 4,165	3,782
Italy	,261	1,265	1,221
Netherlands	11,552	1,500	(*c*) 1,500
Poland (*a*)	10,236	9,305	9,651
Spain	841	868	877
Anglo-Australian	6,261	6,346	6,385
Elsewhere (*b*)	8,900	8,400	8,500
Totals	81,843	80,714	80,887

(*a*) Includes salable zinc dust. (*b*) Partly estimated ;ncludes Norway, Yugoslavia, Czechoslovakia, Russia ıı do-China and Japan. (*c*) Estimate.

Lead Stocks Increase

Price shading in lead took place during the month to revive buying interest. Consumers lost some of their confidence in lead, owing to the recent gain in stocks, and purchases declined in February contrasted with the first month of the year. Stocks of refined lead in the United States on Feb. 1 amounted to 160,257 tons, against 151,380 tons a month previous, according to the American Bureau of Metal Statistics. On the last day of February lead was quoted at 3.50c., New York, and 3.30c., St. Louis. (On March 1 the market fell to 3.25c., New York, and 3.05c., St. Louis.) Corroders were fairly active buyers of lead on the decline.

Enactment of a duty on lead by Great Britain brought the old lead pool to a sudden end. Broken Hill interests withdrew from the organization. A new all-British pool is to be formed.

Tin Visible Increases

Tin prices moved within narrow limits in the last month, with the undertone, in the main, steady. The tin pool has supported the market on several occasions. Higher sterling exchange also was a factor in steadying prices. The so-called world's visible supply of tin on the last day of February was estimated at 51,300 long tons, against 50,043 long tons a month previous. On the strength

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft

All Figures Except Tin Represent Tons of 2,000 Lb.

	1931											1932
COPPER — North and South America	Feb.	March	April	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Jan.
Production, refined, daily average	3,566	2,292	3,350	3,313	3,276	3,110	2,909	2,890	.....	.....	.....	.....
Domestic shipments	60,636	75,685	54,567	45,265	50,217	43,144	45,816	40,459	.....	.....	.....	.....
Foreign shipments	39,415	36,797	32,218	26,684	33,251	26,321	29,016	22,124	.....	.....	.....	.....
Stocks, blister and refined	566,853	553,016	561,797	589,245	600,827	620,075	631,880	658,321	.....	.....	.....	.....
LEAD — United States												
Production, refined, from domestic ore	39,464	41,775	35,498	39,519	30,708	32,157	34,144	31,966	36,546	31,671	33,576	32,180
Production, secondary and foreign	4,654	3,025	2,941	3,598	3,683	4,409	5,453	3,934	4,722	3,820	4,031	4,559
Production, total, daily rate	1,575	1,448	1,280	1,390	1,146	1,179	1,151	1,163	1,331	1,180	1,213	1,185
Shipments reported	34,439	36,761	35,324	34,081	37,054	42,219	38,590	38,059	34,276	31,216	30,297	27,867
Stocks, end of month	122,826	130,426	132,993	142,370	139,698	133,958	134,977	132,804	139,796	144,057	151,380	160,257
ZINC — United States												
Production, daily average	1,056	1,043	971	971	783	689	692	712	699	684	709	723
Domestic shipments	30,249	35,224	27,418	25,831	27,604	28,460	23,599	20,902	21,163	20,327	23,005	22,424
Export shipments	.....	.....	.....	20	.....	20	.....	.....	.....	.....	.....	31
Stocks, end of month	144,389	141,493	143,212	143,049	138,928	131,833	129,701	130,155	130,666	130,865	129,825	129,886
World Production Rate (Daily Average)												
Copper	4,596	4,418	4,296	4,209	4,224	3,919	3,924	4,023	.....	.....	.....	.....
Lead	4,883	4,693	4,898	3,988	3,995	3,691	3,906	3,654	3,806	3,980	3,958	3,906
Zinc	3,721	3,574	3,429	3,212	3,127	2,860	2,766	2,815	2,740	2,728	2,604	2,609
Tin (Metric Tons)	504	436	441	426	439	401	416	349	352	325	360	.....

of estimates of smaller Straits shipments, traders expected a decline in stocks.

Silver Higher

Offerings of silver were moderate and, with speculative operators inclined to the long side of the market, prices realized were somewhat higher than in January. The market settled at 30⅞c., New York, against 29¾c. a month ago. United States production of silver during January was 2,032,000 oz., against 2,350,000 oz. in December, and 3,480,000 oz. in January of last year.

Lead and Zinc Concentrates

Owing to the decline in the metal, the Joplin, Mo., market for lead concentrate, basis 80 per cent, dropped to $32.50@$37.50 per ton. A weekly average of 881 tons of lead concentrate has been shipped so far this year, against 612 tons a week in the same period last year.

Zinc, concentrate, Prime, basis 60 per cent, settled at $18 per ton in the Tri-State district, against $17 a ton a month ago.

Other Metals

Quotations cover wholesale lots, f.o.b. New York, unless otherwise specified. London prices are according to latest mail advices, and at present are largely nominal in view of erratic sterling exchange.

ALUMINUM—Per lb., delivered, Alcoa commercial and mill ingot, 99 and 98 per cent, 23.30c.; Alcoa No. 12 alloy, 22c.; metallurgical ingot, 94 per cent plus, 23.30c.; 98@99 per cent, 22.90c. London, £95, less 2 per cent, per long ton, for 98 per cent ingots and bars.

ANTIMONY—Per lb., duty paid. Chinese and American, spot, 6.375c.; futures, nominal at 6.25c.

BERYLLIUM—See Alloys.

BISMUTH—Per lb., in ton lots, 85c. London (dollar basis), 75c.

CADMIUM—Per lb., 55c. London, 2s. 5d. to 2s. 6d. Nominal.

CHROMIUM—Per lb., 97 per cent grade, spot, 88c.; contract 83c. per lb. contained chromium, maximum 1 or 2 per cent iron. (Usually sold as ferrochrome.)

COBALT—Per lb.: Metal imported from Belgium, 97@99 per cent, $2.50 less 35 per cent for cash. On yearly requirements, usual rebate of 5 to 10 per cent, as to quantity. London quotes $3.65 per kilo, subject to usual discounts.

INDIUM—Per oz., 98½ per cent, $90@$100 per oz. Nominal.

IRIDIUM—Per oz., $80@$90 for 98@99 per cent sponge and powder. London, £24@£26. Nominal.

LITHIUM—Per lb., 98@99 per cent, 100-lb. lots, $15.

MAGNESIUM—Per lb., ingots (4x16 in.), 99½ per cent, 30c. in carloads; 32c. in 100 lb. lots or more, l.c.l.: ⅛, ¼, 1, and 2 lb. sticks, 5c. per lb. over ingot price; 1/10 and ¼ lb. sticks, 8c. per lb. extra.

MANGANESE—Per lb., 95@97 per cent, 42c. (Usually sold as ferromanganese.)

MOLYBDENUM—Per lb., in 10- to 50-lb. lots, C.P. powder, $9; 97 per cent, $4.50. (Usually sold as calcium molybdate or ferromolybdenum, which see.)

NICKEL—Per lb. electrolytic cathodes, 35c.; shot and ingot made from remelted electro, 36c. for single lots of spot metal. London, per long ton, £245 to £250, with prices nominal.

OSMIUM—Per oz., $65@$70. London, £20 10s. to £21.

PALLADIUM—Per oz., $19@$21. London, £5 5s., nominal.

PLATINUM—Per oz. Official price of leading interest, $40. Cash transactions between dealers and refiners several dollars less. London market nominal at £10 18s. to £11 4s. The average price of Colombian crude, basis 85 per cent platinum, for the month of January, was $29.55 per oz.

QUICKSILVER—Per 76-lb. flask, $67.50. The price advanced during February on smaller offering by producers. Demand was quiet.

RADIUM—Per mg. radium content, $50 in lots of 4 grams or more, to $65 for 1 gram; smaller quantities, $70.

RHODIUM—Per oz., $35@$50.

RUTHENIUM—Per oz., $38@$48.

SELENIUM—Per lb., $1.80@$2, depending on quantity, for black, powdered, 99.5 per cent pure.

SILICON—Per lb., minimum 97 per cent Si, maximum 1 per cent Fe, spot, 16c.; contract, 14c.

TANTALUM—Per kilo, $91 for C.P. bar or sheet.

TELLURIUM—Per lb., $2.

THALLIUM—Per lb., $12.50@$15.

TITANIUM—Per lb. in 50- to 100-lb. lots, 75 per cent, $5.

TUNGSTEN—Per lb., contained tungsten, 98 per cent, powdered, $1.45.

Average Metal Prices for February, 1932

COPPER:	
Electrolytic, refinery	5.965
London, Standard, Spot	36.917
London, Electrolytic, Bid	41.381
LEAD:	
New York	3.712
St. Louis	3.499
London, Spot	14.560
London, Forward	14.571
SILVER:	
New York	30.136
London	19.573
Sterling Exchange	345.141
ZINC:	
St. Louis	2.817
London, Spot	13.872
London, Forward	14.571
TIN:	
Straits	22.018
London, Standard Spot	139.143
QUICKSILVER	66.304
ANTIMONY	6.489
PLATINUM, Refined	40.000
CADMIUM	55.000
ALUMINUM, 99 Per Cent Plus	23.300

■

Metallic Ores

Prices in tons of 2,000 lb., or in "units" of 20 lb., unless otherwise stated.

ANTIMONY ORE—No sales reported. London, per long ton unit, 4s. 6d. for 60 to 65 per cent sulphide ore.

BERYLLIUM ORE—Per ton, minimum 10 per cent BeO, $20@$30, f.o.b. mines.

CHROME ORE—Per long ton, c.i.f. Atlantic ports, Indian ores, $17.50 for 48@50 per cent Cr_2O_3 ore, and $19.50@$20 for 50@51 per cent ore. London, 80s. for 48 per cent Rhodesian.

COBALT ORE—Per lb. of Co, 12 to 14 per cent grade, 50c., f.o.b. cars, Ontario. Market nominal.

IRON ORE—Per long ton, Lower Lake ports. Lake Superior ores:

Mesabi, non-bessemer, 51½ per cent iron, $4.50. Old Range, non-bessemer, $4.65.

Mesabi, bessemer, 51½ per cent iron, $4.65. Old Range, bessemer, 51½ per cent, $4.80.

Eastern ores, cents per long ton unit, delivered at furnaces: Foundry and basic, 56 to 63 per cent, 10c.

Foreign ores, alongside docks Atlantic ports, cargo lots, cents per long ton unit:

North African and Swedish, low-phosphorus, 8@9c.

Spanish and North African, basic, 56@60 per cent, 7@8½c.

Swedish foundry or basic, 65@68 per cent, 8c.

Newfoundland foundry, 55 per cent iron, 8c. Nominal.

MANGANESE ORE—Per long ton unit of Mn, c.i.f. North Atlantic ports, cargo lots, exclusive of duty: Brazilian, 46@48 per cent Mn, 21c.; Chilean, 47 per cent minimum, 24, nominal; Indian, 48@50 per cent, 24c.; Caucasian, 52@55 per cent, 24c.; South African, 52@54 per cent, 23@24c.; 50 to 52 per cent, 22@23c.; 44 to 46 per cent, 21c.

Per ton in carload lots:

Chemical grades, powdered, coarse or fine, minimum 80 per cent MnO_2, Brazilian or Cuban, $50 in carloads, to $60 barreled. Javan or Caucasian, 85 per cent minimum, $50@$60. Domestic, 70 to 72 per cent, $43@$50 in carloads, f.o.b. mines.

MOLYBDENUM ORE—Per lb. of contained MoS_2, nominally 42c., delivered Pittsburgh, for 75@85 per cent concentrate. London, per long ton unit, nominal at 37s. to 38s. for 80@85 per cent concentrate.

TANTALUM ORE—Per lb., Ta_2O_5, 70c. for 60 per cent ore. Market nominal.

TIN ORE — No market in United States. London quotes £9 to £9 10s. per ton returning charges on 60 per cent Bolivian ore.

TITANIUM ORE — Per gross ton, ilmenite, 45@52 per cent TiO_2, f.o.b. Atlantic seaboard, $10@$12, according to grade and impurities. Low-grade domestic, 32 to 35 per cent, about $7 @$8. Rutile, per lb., guaranteed minimum 94 per cent concentrate, 10c.

TUNGSTEN ORE—Per unit of WO_3, N. Y.: Chinese wolframite, $10.75@ $11, duty paid. Bolivian scheelite, $10.75. Domestic, $10@$10.50.

VANADIUM ORE—Per lb. V_2O_5 contained, 27c., f.o.b. shipping point.

ZIRCON ORE—Per ton, 55 per cent ZrO_2, f.o.b. Atlantic seaboard, $40@$45 in 30-ton lots. Crude granular zircon, $70, f.o.b. Suspension Bridge, N. Y.; milled, $90.

Non-Metallic Minerals

Prices received for non-metallic minerals vary widely and depend upon the physical and chemical characteristics of the commodity. Hence the following quotations can serve only as a general guide to the prices obtained by producers and dealers in different parts of the United States for their own product. In the last analysis the value of a particular non-metallic mineral can be ascertained only by direct negotiation between buyer and seller.

Tons of 2,000 lb. unless otherwise noted.

AMBLYGONITE — Per ton, f.o.b. mines, 8@9 per cent Li_2O, $50@$60.

ASBESTOS — Per ton, f.o.b. Quebec mines, tax and bags included: Crude No. 1, $400 (nominal); crude No. 2, $200; spinning fibers, $75@$125; Magnesia and compressed sheet fibers, $70@$85; various grades shingle stock, $45@$60; various grades paper stock, $27.50@$35; cement stock, $15@$20; floats, $10@$15.

Per ton c.i.f. New York: Rhodesian crude No. 1, $250; No. 2, $175. Nominal.

Per ton, c.i.f. New York: Russian crude No. 1, $200@$225; No. 2, $150@ $175; No. 3, $125. Nominal.

Per ton, f.o.b. mines, Vermont: Shingle stock, $40@$45; paper stock, $30@$35; cement stock, $18@$22.

BARIUM CARBONATE (Witherite) — Per ton, 90 per cent 300 mesh, $42; 100 mesh, $39.

BARYTES—F.o.b. mines:

California: Crude, $7 per ton.

Georgia: Barytes ore, crude, $6@ $6.50 per long ton.

Missouri: Per ton, water ground and floated, bleached, $23, car lots, f.o.b. works. Crude ore, minimum 95 per cent $BaSO_4$, less than 1 per cent iron, $6; 1 per cent iron and 93 per cent $BaSO_4$, $5.50; low-grade, $4.50, f.o.b. mines.

BAUXITE—Per long ton: Domestic ore, chemical, crushed and dried, 55 to 58 per cent Al_2O_3, 1.5 to 2.5 per cent Fe_2O_3, $6.50@$8, f.o.b. *Alabama* and *Arkansas* mines. Other grades, 56 to 59 per cent Al_2O_3, 5 to 8 per cent SiO_2, $6@$7.50, f.o.b. *Arkansas* mines. Pulverized and dried, 56 to 59 per cent Al_2O_3, 8 to 12 per cent SiO_2, $10@ $12.25, f.o.b. *Arkansas* mines; abrasive grade, crushed and calcined, 78 to 84 per cent Al_2O_3, $14@$16, f.o.b. *Arkansas* mines.

Per metric ton, foreign, c.i.f. Atlantic ports: Dalmatian, 50 to 55 per cent Al_2O_3, 1 to 3 per cent SiO_2, $4.50@ $6. Istrian, 54 to 57 per cent Al_2O_3, 3 to 5 per cent SiO_2, $5.50@$6.50; French, 56 to 59 per cent Al_2O_3, 2 to 4 per cent SiO_2, $5.50@$7.

BENTONITE—Per ton, carload lots, f.o.b. *Wyoming* mines, dried and crushed, in bulk, $8; in bags, $10. F.o.b. Chicago, selected air-floated, $25.

BORAX — Per ton, carload lots, in bags, crystals, $56; granulated, $50; powdered, $57.50; f.o.b. shipping point.

CELESTITE—Per ton in carload lots, 90 per cent $SrSO_4$, finely powdered, $27.

CHINA CLAY (Kaolin)—Per ton, f.o.b. *South Carolina* mines, crude lump, No. 1, $4@$5; crushed, $6.90; pulverized, $9@$15.

Florida, washed and crushed, No. 1, $12.50; No. 2, $12.

Delaware, No. 1, $14.50.

Imported English, f.o.b. American ports: lump, $17@$21 in bulk.

DIATOMITE—Per ton, f.o.b. *Nevada,* crude, dried, in bags, $10; minus 40 mesh, $15; 300 mesh, $20; high temperature insulation, $30.

EMERY—Per ton, f.o.b. *New York,* domestic crude ore, first grade, $10. Other American ore, delivered to grinders, per gross ton, $16; Turkish and Naxos ore, $30@$35. F.o.b. *Pennsylvania,* in 350-lb. kegs, Turkish, Khasia, and Naxos grain emery, 6½c. per lb.; American, 3½c.

FELDSPAR — Per ton, f.o.b. *North Carolina,* potash feldspar, 200 mesh, white, $15, in bulk; soda feldspar, $18. F.o.b. *Maine,* potash feldspar, white, 200 mesh, $19, in bulk. Granular glasspar, white, 20 mesh, f.o.b. *North Carolina,* $10.50 in bulk.

Virginia, No. 1, 325 mesh, $17; 200 mesh, $15@$16; 160 mesh, $14@$15; No. 1 glassmakers', $9.50, Spruce Pine basis; enamelers', $12.50.

New Mexico: Crude clean No. 1 potash spar, $4.75; ground, $9.50.

FLUORSPAR—Per net ton, 85 per cent CaF_2, and not over 5 per cent SiO_2. *Kentucky* and *Illinois* mines, contract, freight allowed Pittsburgh basis: Washed gravel, $20.31; No. 2 lump, $22.31.

Ground fluorspar, f.o.b. *Illinois* mines, 95 to 98 per cent CaF_2 and not over 2½ per cent SiO_2, $30 in bulk; $34 in bags or barrels.

F.o.b. *Colorado* mines, 82-5, $10.

Foreign fluorspar, gravel, 85-5, $18@ $19 per gross ton, duty paid.

FULLER'S EARTH — Per ton, f.o.b. *Colorado,* $18, various finenesses to minus 90 mesh.

F.o.b. *Georgia* or *Florida,* 30 to 60 mesh, $16; 15 to 30, $15; 200 up, $10; 100 up, $7.

GARNET—Per ton, f.o.b. *New Hampshire* mines; concentrate, $40; washed grades, $125.

New York: Adirondack garnet concentrates, $85.

Spanish grades, $60, c.i.f. port of entry.

GILSONITE — Per ton, carload lots, f.o.b. *Colorado.*

Brilliant black, $32.90; seconds (mine run), $25.50.

Selects, $30.50, f.o.b. Utah. Nominal.

GRAPHITE—Per lb., f.o.b. *New York:* Ceylon lump, 6½@7½c.; carbon lump, 3@6c.; chip, 5@6c.; dust, 3@4c.; Madagascar flake, 5@6c.

No. 1 flake, 8@16c.; No. 2, 5½c. upward; fine ground, 55 to 70 per cent carbon, 3c. upward; amorphous, 3c. upward.

Crude amorphous graphite, $12@$23 per ton, according to grade.

GREENSAND — Per ton, f.o.b. cars, *New Jersey:* Screened and bagged, best grade, in carload lots, $20.

GYPSUM—Per ton, f.o.b. mill, crushed rock, *Iowa,* $2.50; *Ohio* and *New York,* $3. Ground, *Ohio,* $4; *Iowa,* $6. Agricultural, $6@$7. Calcined, $6@$9.

IRON OXIDE (See Ocher)—Per lb.: Standard Spanish red, 3@4½c.; domestic earth, 2@4½c.

KAOLIN—See China Clay.

KIESELGUHR—See Diatomite.

LEPIDOLITE—Per ton, $50@$60 for ordinary grades. Nominal.

LIMESTONE—Per ton, f.o.b. shipping points, depending on location, either lump or crushed, 25c.@$1.75.

Agricultural, $1 up to $6 per ton, as to grade and point of shipment.

MAGNESITE—Per ton, f.o.b. *California,* dead-burned, $25. Kiln run, 93 per cent MgO (artificial periclase), $68; 88 per cent MgO, $35. Caustic, 95 per cent MgO, $45; 90 per cent, $40. *Washington:* Dead-burned grain magnesite, $22.

MICA—Per ton, f.o.b. *New Mexico,* scrap, white, $21; off color, $18. Punch, white, for disks, per lb., 10c.; for washers, 8c.

Per ton, f.o.b. *New Hampshire,* roofing mica, $23; snow, $34; 40 mesh white, $40; 60 mesh, $48; 100 mesh, 60; 200 mesh, $75. Clean dry mixed bench and mine scrap, $13.

Per lb., f.o.b. *North Carolina:* Punch, 3@5c.; 1½x2 in., 20@30c.; 2x2, 30@50c.; 2x3, 60@90c.; 3x3, $1 @$1.40; 3x4, $1.40@$1.80; 3x5, $1.50 @$2; 4x6, $2.25@$3; 6x8, $2.50@ $3.75; 8x10, $5@$7.50. These prices apply both to No. 1 and No. 2 mica. Black-stained mica takes a discount of 10 to 25 per cent from this schedule. White North Carolina mica, 70 mesh, $60 to $80. Biotite, or black mica, $15 a ton, unground. White, Georgia, 300 mesh, $19.50; sericite, 300 mesh, $15 per ton.

MINERAL BLACK—(Graphitic shale)—Per ton, 325 mesh, 75 per cent carbon. $22.50, f.o.b. *Virginia.*

MONAZITE—Per ton, minimum 6 per cent ThO_2, $60.

OCHER—Per ton, f.o.b. *Georgia* mines, $17.50 in sacks; $21 in barrels. Buff clay, 98 per cent through 325 mesh, $18.

F.o.b. *Virginia,* dark yellow, 300 mesh, 60 per cent ferric oxide, in jute bags, $19.50.

PHOSPHATE—Per long ton, f.a.s. or f.o.b. mines:

Florida, pebble, f.a.s., for export: 76@77 per cent, $7.15; 75 per cent, $6.40; 74@75 per cent, $6.15; 70 per cent, $4.65; 68 per cent, $4.10.

Florida, pebble, domestic: 76@77 per cent, $6.50; 74@75 per cent, $5.50; 72 per cent, $4.50; 70 per cent, $4.

Tennessee, ground lime phosphate, 80 per cent through 300 mesh, 33 per cent P_2O_5, $12.80 per net ton, bags extra. Rock phosphate, 29 per cent P_2O_5, 200 mesh, $9.50. Furnace lump $6.25.

PUMICE STONE—Per lb., in barrels, powdered, 2½@4c.; lump, 5@7c.

PYRITES—Per long ton unit of sulphur, c.i.f. United States ports, guaranteed 48 per cent sulphur, Spanish, 13c.

POTASH—Per ton, contracts:

	Bags	Bulk
Muriate of potash, 80@85 per cent, basis 80 per cent	$37.15	$35.55
Sulphate of potash, 90@95 per cent, basis 90 per cent	48.25	46.65
Sulphate of potash-magnesia, 48@53 per cent, basis 48 per cent	27.80	26.20
Manure salt, 30 per cent	22.15	19.15
Manure salt, 20 per cent	15.65	12.65
Kainit, 14 per cent	12.70	9.70

Prices are delivered to Atlantic ports. Discounts up to 4 per cent for early shipment. Spot prices, $1.25@$1.50 per ton higher than on contract.

[1]QUARTZ ROCK CRYSTALS—For fusing, all sizes, $250 per ton. Prisms for piezo-electrical and optical use command premium.

SILICA—Per ton, water ground and floated, in bags, f.o.b. *Illinois:* 325 mesh, $16@$40 for 92 to 99½ per cent grades. Dry ground, air-floated, 325 mesh, 92@99½ per cent silica, $14@$30. Glass sand f.o.b. producing plant, $1.25@$5 per ton; molding sand, 50c.@$3.50; blast sand, $1.75@$6. *California:* $5 for quartz and $2.50 for sand.

[1]STRONTIANITE—Per ton, lump in carload lots, minimum 90@92 per cent $SrCO_3$, $100.

SULPHUR—Per long ton for domestic market, $18 f.o.b. Texas mines.

TALC—Per ton, carload lots, f.o.b. works, containers included unless otherwise specified:

New Jersey: Soapstone, ground, $10@$12.

New York: Double air floated, short fiber, 200 mesh, $13.75; 325 mesh, $14.75.

Vermont: 99 per cent through 200 mesh, extra white, bulk basis, $7.50@$8; 97 to 98 per cent through 200 mesh, medium white, $6.50@$7. Packing in paper bags, $1 per ton extra.

Virginia: 200 mesh, $4.60@$5.50, 325 mesh, $8@$9.25.

TRIPOLI — Per ton, burlap bags, paper liners, minimum carload 30 tons, f.o.b. *Missouri:* Once ground through 40 mesh, rose and cream colored, $15. Double ground through 110 mesh, rose and cream, $17. Air-floated through 200 mesh, rose and cream, $25.

WHITING—Per ton, f.o.b. *Georgia,* white, 300 mesh, $7.

Metallic Compounds

ARSENIOUS OXIDE (White arsenic)—Per lb., 4c., delivered, all positions.

CALCIUM MOLYBDATE OR MOLYTE—Per lb. of contained Mo, 85c.

COBALT OXIDE—Per lb., black oxide, 70@71 per cent grade, $1.35.

COPPER SULPHATE (Blue Vitriol)—Per lb., in car lots, 2.75c. for either large or small crystals.

SODIUM NITRATE—Per 100 lb., crude natural, in bags ex vessel, Atlantic ports, $1.77.

SODIUM SULPHATE (Salt Cake)—Per ton, bulk, f.o.b. works, $16@$18.

ZINC OXIDE—Per lb. in bags, in car lots: Lead-free, 6½c.; 10, 20 or 35 per cent leaded grades, 6¼c.; French red seal, in bags, 9⅜c.

ZIRCONIUM DIOXIDE—Per lb., carload lots, commercial, 38@43c.

Refractories

CHROME BRICK—Per ton, f.o.b. shipping point, $42.50. Grecian refractory ore, 37@42 per cent, 40c. per unit.

FIRECLAY BRICK—Per M., first quality, $38, *Ohio, Kentucky, Central Pennsylvania;* second quality, $30.

MAGNESITE—Brick, per ton, f.o.b. works, 9-in. straights, $61.50. Dead-burned grain, $38.50, f.o.b. Chester, Pa., or Baltimore.

SILICA BRICK—Per M., *Pennsylvania,* $38; *Alabama,* $50; *Illinois* and *Indiana,* $47.

ZIRKITE—Per lb., powdered, 65@70 per cent ZrO_2, 3½c. Brick, straights, 80c.@$1 each.

Rolled Metals

COPPER—Per lb., sheets, hot-rolled, 15⅜s.; wire, f.o.b. mill 8c., nominal.

LEAD SHEETS—Per lb., full rolled, 6¾c.; clipped, 7c.

MONEL METAL—Per lb., sheets, full finished, 42c.; strip, cold-rolled, 45c.; rods, hot-rolled, 35c.

MUNTZ METAL—Per lb. rods, 11¼c.; sheets, 14⅝c.

NICKEL—Per lb., sheets, full finished, 52c.; rods, hot-rolled, 45c.

NICKEL SILVER—Per lb., sheets, 10 per cent, 20⅝@21c.; 18 per cent, 24⅛@24½c. Wire and rods, 10 per cent, 24½c.; 15 per cent, 28⅛c.; 18 per cent, 31⅝c.

PHOSPHOR BRONZE—Per lb., sheets and rods, 5 per cent tin, 26½c.; wire, 5 per cent, 26⅝c.; 10 per cent, 31½c.

ZINC SHEETS—Per lb., 9c., f.o.b. works; ribbon, 8½c.; 7 per cent discount on orders for 18 tons or more.

Alloys

BERYLLIUM-COPPER—Per lb., containing 12.5 per cent of Be, $6.35.

FERROCHROME—Per lb. of contained chromium, 4 to 6 per cent carbon, 65 to 70 per cent chromium, 10c., delivered, on contracts; spot, 10½c. Containing 2 per cent carbon, 67 to 72 per cent chromium, 17c.; maximum 0.10 per cent carbon, 23½c.

FERROMANGANESE—Per gross ton, furnace: domestic and foreign, 78@82 per cent, $72@$75, contracts, depending on quantity. Spiegeleisen, 19@21 per cent, $27 for spot carloads.

FERROMOLYBDENUM—Per lb. of Mo, f.o.b. shipping point, 50@60 per cent Mo, 95c.

FERROPHOSPHORUS—Per gross ton, 18 per cent P, $91; electrolytic, 24 per cent, $122.50, f.o.b. *Alabama* and *Tennessee.*

FERROSILICON—Per gross ton, delivered, 50 per cent, $77.50; 75 per cent, $126; 14@16 per cent, $31, f.o.b. Niagara Falls, N. Y. Spot shipments $5@$10 per ton higher on 50 and 75 per cent grades.

FERROCARBONTITANIUM — Per ton, $140, f.o.b. producer's plant, carlots.

FERROTUNGSTEN—Per lb. of W contained, 75@80 per cent W, $1@$1.10, f.o.b. works.

FERROVANADIUM—Per lb. of V contained, delivered, $3.05@$3.30.

SILICOMANGANESE — Per long ton, carload contracts, delivered, $105 for maximum 2½ per cent carbon; $115 for maximum 1 per cent carbon.

SILICON ZIRCONIUM—Per lb., 47@52 per cent Si, 35@40 per cent Zr, 16@19c.

ZIRCONIUM FERROSILICON—Per gross ton, 12@15 per cent Zr, 39@43 per cent, Si, $97.50@$105.

Iron, Steel, and Coke

IRON—Per gross ton, Valley furnaces: Bessemer, $15.75@$16; basic, $15; and No. 2 foundry, $15.50@$16.

STEEL—Per gross ton, base prices, Pittsburgh, billets and slabs, $27; bars, plates, and structural shapes, per 100 lb., $1.50.

COKE—Per ton, Connellsville furnace, $2.25@$2.35. Connellsville foundry, $3.25@$4.25. Birmingham furnace, $3.50@$4.50. Byproduct coke, Ohio and Kentucky (Connellsville basis), $5.50; Buffalo and Detroit, $8@$8.50; Birmingham foundry, $5.

[1]Price furnished by Foote Mineral Company, Philadelphia.

Daily Prices of Metals

For the month of February, 1932

United States Market

Feb.	Electrolytic Copper, Refinery	Straits Tin New York	Lead — New York	Lead — St. Louis	Zinc St. Louis
1	6.525	22.050	3.75	3.425@3.55	2.825
2	6.275@6.525	22.125	3.75	3.55	2.80 @2.825
3	6.275	21.850	3.75	3.50 @3.55	2.80 @2.825
4	6.025@6.275	21.95 @22.00	3.75	3.55	2.80
5	6.025@6.275	21.875@21.90	3.75	3.55	2.80
6	5.775@6.025	21.875@21.90	3.75	3.55	2.80
8	5.775	21.750	3.75	3.55	2.80 @2.85
9	5.775	21.625@21.70	3.75	3.55	2.80 @2.90
10	5.775	21.400	3.75	3.55	2.80
11	5.775	21.700	3.75	3.50 @3.55	2.80
12			Holiday		
13	5.775	22.000	3.75	3.50 @3.55	2.80
15	6.025	22.250	3.75	3.50 @3.55	2.80 @2.85
16	6.025	22.125	3.75	3.55	2.825
17	6.025	22.250	3.75	3.55	2.825
18	6.025	22.20 @22.25	3.75	3.55	2.825@2.85
19	6.025	22.25 @22.30	3.75	3.55	2.825@2.85
20	6.025	22.25 @22.30	3.75	3.55	2.825@2.85
22			Holiday		
23	6.025	22.10 @22.25	3.75	3.50	2.825@2.85
24	.775@6.025	22.15 @22.20	3.75	3.475@3.50	2.825@2.85
25	5.775	22.125	3.50@3.75	3.30 @3.50	2.80
26	5.775	22.00 @22.10	3.50	3.30	2.80
27	5.775	22.100	3.50	3.30	2.80
29	5.525	22.100	3.50	3.30	2.80
Av. for month	5.965	22.018	3.712	3.499	2.817
Feb.		Averages for Week			
3	6.525	21.888	3.750	3.535	2.850
10	5.921	21.760	3.750	3.550	2.813
17	5.925	22.065	3.750	3.535	2.815
24	6.000	22.225	3.750	3.528	2.838
Feb.		Calendar Week Averages			
6	6.233	21.963	3.750	3.535	2.808
13	5.775	21.703	3.750	3.540	2.815
20	6.025	22.233	3.750	3.546	2.831
27	5.850	22.125	3.625	3.398	2.815

Silver, Gold, and Sterling Exchange New York and London

Feb.	Sterling Exchange "Checks"	Sterling Exchange "90-Day" Demand	Silver — New York	Silver — London	Gold, London
1	344.25	344.250	30.250	20.0000	120s 2d
2	345.00	345.000	29.875	19.6250	119s 7d
3	344.75	344.750	29.500	19.2500	119s 5d
4	344.50	344.500	29.625	19.2500	120s 1d
5	344.75	344.750	29.750	19.3750	119s 9d
6	344.00	344.000	29.625	19.2500	.....
8	344.50	344.500	29.625	19.2500	120s 1d
9	344.00	344.000	29.625	19.3750	119s 11d
10	342.25	342.375	29.500	19.3125	120s 4d
11	343.00	343.125	29.625	19.3125	120s 9d
12	Holiday	Holiday	Holiday	19.5000	120s 3d
13	344.25	344.500	30.125	19.5000	.....
15	345.00	345.500	30.375	19.6250	119s 11d
16	344.25	344.750	30.250	19.6250	120s 1d
17	344.25	344.750	30.000	19.5625	120s 1d
18	344.25	344.750	30.000	19.5625	120s 1d
19	344.75	345.125	30.125	19.6250	120s
20	344.75	345.125	30.375	19.7500	.....
22	Holiday	Holiday	Holiday	19.8125	120s
23	346.75	347.000	31.000	19.9375	119s 1d
24	347.50	347.875	31.000	19.8125	118s 9d
25	347.75	348.125	30.750	19.8125	118s 10d
26	347.75	348.250	30.500	19.6875	118s 10d
27	348.00	348.625	30.750	19.7500	.....
29	348.00	349.500	30.875	19.7500	118s 5d
Av. for month	345.141	.....	30.136	19.573	.....
Feb.			Averages for Week		
3	344.958	.....	29.646	.....	.....
10	344.000	.....	29.625	.....	.....
17	344.150	.....	30.075	.....	.....
24	345.600	.....	30.500	.....	.....

Calendar week averages: New York Silver, February 6th, 29.771c.; 13th, 29.700c.; 20th, 30.188c.; 27th, 30.800c.

London Market

Feb.	Copper — Standard — Spot	Copper — Standard — 3 Mo.	Copper — Electrolytic (Bid)	Tin — Spot	Tin — 3 Mo.	Lead — Spot	Lead — 3 Mo.	Zinc — Spot	Zinc — 3 Mo.
1	37.6250	37.6250	45.00	139.750	142.375	14.5000	14.6250	14.0000	14.3750
2	38.1875	38.0625	43.50	140.875	143.500	14.6250	14.7500	14.1875	14.5625
3	37.8750	37.5000	43.50	138.625	141.250	14.3750	14.5000	13.8750	14.3125
4	37.6250	37.5625	43.50	139.125	141.750	14.4375	14.5000	13.8125	14.3125
5	37.7500	37.6250	43.00	139.500	142.250	15.0000	15.0000	13.8750	14.3125
8	36.2500	36.1250	42.00	138.250	141.125	14.5000	14.6250	13.8750	14.2500
9	36.3125	36.1875	42.00	138.250	141.000	14.5000	14.6250	13.8750	14.3125
10	35.7500	35.2500	40.00	137.000	139.875	14.5625	14.6875	13.7500	14.1875
11	35.3125	35.1875	39.25	136.125	138.875	14.6875	14.8125	13.5625	14.0625
12	36.7500	36.5625	41.00	138.125	140.875	14.8125	14.8125	14.0000	14.4375
15	38.5000	38.2500	41.00	140.625	143.500	15.0000	15.0625	14.3125	14.7500
16	37.8750	37.3125	42.00	139.500	142.250	14.9375	14.9375	14.3750	14.7500
17	38.7500	37.8750	42.00	140.500	143.375	15.0000	15.0000	14.4375	14.7500
18	38.8750	38.2500	41.50	140.750	143.375	15.0000	15.0000	14.3750	14.7500
19	38.5000	37.9375	41.00	140.125	142.750	14.9375	15.0000	14.2500	14.6250
22	38.3750	37.3750	41.00	140.625	143.125	14.7500	14.7500	14.1250	14.5000
23	36.8750	36.3750	41.00	139.750	142.125	14.3750	14.3125	13.7500	14.1250
24	35.6875	35.8125	40.50	139.250	141.500	14.1250	13.8750	13.2500	13.7500
25	35.2500	35.3750	39.50	139.000	141.375	14.2500	14.0000	13.2500	13.8125
26	33.3750	33.5000	38.50	137.750	140.125	13.8750	13.7500	13.3750	13.7500
29	33.7500	34.1250	38.25	138.500	140.750	13.5000	13.3750	13.0000	13.3750
Average for mo.	36.917	.....	41.381	139.143	.....	14.560	14.571	13.872	14.289

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices are in cents per pound.

Copper, lead, and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are 1c. above St. Louis prices for Prime Western.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, basis of 925 fine. Sterling quotations represent the demand market in the forenoon. Cables command a premium.

Monthly Average Prices of Metals

(See daily price table for basis of quoting)

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1931	1932	1931	1932	1931	1932
January	29.423	29.780	13.810	19.623	485.260	342.515
February	26.773	30.136	12.432	19.573	485.551	345.141
March	29.192	.....	13.524	.....	485.596	.....
April	28.279	.....	13.120	.....	485.764	.....
May	27.650	.....	12.858	.....	486.188	.....
June	27.250	.....	12.707	.....	486.291	.....
July	28.255	.....	13.197	.....	485.349	.....
August	27.524	.....	12.815	.....	485.284	.....
September	28.180	.....	14.101	.....	451.245	.....
October	29.538	.....	17.153	.....	388.029	.....
November	32.223	.....	19.393	.....	371.130	.....
December	30.120	.....	20.023	.....	336.798	.....
Year	28.700	.....	14.594	.....	452.707	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery		London Spot			
	Electrolytic		Standard		Electrolytic	
	1931	1932	1931	1932	1931	1932
January	9.838	7.060	44.938	39.459	47.524	46.200
February	9.724	5.965	45.372	36.917	47.950	41.381
March	9.854	.....	44.818	.....	47.699	.....
April	9.392	.....	42.694	.....	45.375	.....
May	8.665	.....	38.897	.....	42.175	.....
June	8.025	.....	35.827	.....	38.966	.....
July	7.698	.....	34.402	.....	37.293	.....
August	7.292	.....	32.572	.....	35.388	.....
September	6.988	.....	31.503	.....	36.148	.....
October	6.775	.....	34.957	.....	41.000	.....
November	6.558	.....	35.854	.....	41.190	.....
December	6.580	.....	38.273	.....	44.409	.....
Year	8.116	.....	38.342	.....	42.093	.....

New York quotations, cents per pound. London, pounds sterling per ong ton.

Lead

	New York		St. Louis		London			
	1931	1932	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.802	3.750	4.604	3.550	13.872	13.905	15.084	15.128
February	4.552	3.712	4.340	3.499	13.444	13.550	14.560	14.571
March	4.527	.....	4.276	.....	13.128	13.355	.....	.....
April	4.412	.....	4.164	.....	12.375	12.606	.....	.....
May	3.818	.....	3.651	.....	11.491	11.778	.....	.....
June	3.917	.....	3.761	.....	11.582	11.952	.....	.....
July	4.400	.....	4.225	.....	12.731	12.899	.....	.....
August	4.400	.....	4.225	.....	11.944	11.994	.....	.....
September	4.400	.....	4.217	.....	11.932	12.026	.....	.....
October	3.964	.....	3.773	.....	13.227	13.270	.....	.....
November	3.937	.....	3.758	.....	14.577	14.491	.....	.....
December	3.792	.....	3.592	.....	15.188	15.361	.....	.....
Year	4.243	.....	4.049	.....	12.958	13.099	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1931	1932	1931	1932
	Straits		Spot	
January	26.137	21.804	115.798	140.219
February	26.315	22.018	117.919	139.143
March	27.065	.....	121.852	.....
April	25.222	.....	112.775	.....
May	23.221	.....	104.331	.....
June	23.478	.....	104.966	.....
July	24.978	.....	111.478	.....
August	25.738	.....	114.875	.....
September	24.618	.....	117.813	.....
October	22.723	.....	126.932	.....
November	22.779	.....	132.857	.....
December	21.328	.....	138.909	.....
Year	24.467	.....	118.375	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.035	3.011	12.747	13.113	14.416	14.834
February	4.012	2.817	12.303	12.694	13.872	14.289
March	4.002	.....	12.190	12.676	.....	.....
April	3.717	.....	11.353	11.838	.....	.....
May	3.306	.....	10.484	10.875	.....	.....
June	3.416	.....	11.270	11.750	.....	.....
July	3.893	.....	12.280	12.802	.....	.....
August	3.817	.....	11.444	12.028	.....	.....
September	3.744	.....	11.571	12.063	.....	.....
October	3.377	.....	12.733	13.216	.....	.....
November	3.209	.....	13.845	14.247	.....	.....
December	3.149	.....	14.361	14.818	.....	.....
Year	3.640	.....	12.215	12.677	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1931	1932	1931	1932
January	67.115	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	.....	23.300	.....
April	55.000	.....	23.300	.....
May	55.000	.....	23.300	.....
June	55.000	.....	23.300	.....
July	55.000	.....	23.300	.....
August	55.000	.....	23.300	.....
September	55.000	.....	23.300	.....
October	55.000	.....	23.300	.....
November	55.000	.....	23.300	.....
December	55.000	.....	23.300	.....
Year	56.010	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade.
Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (*a*) New York		Quicksilver (*b*) New York		Platinum (*c*) New York	
	1931	1932	1931	1932	1931	1932
January	7.317	5.976	103.000	64.900	36.000	40.000
February	7.069	6.489	100.205	66.304	34.000	40.000
March	7.127	.....	100.423	.....	29.538	.....
April	6.888	.....	102.077	.....	26.346	.....
May	6.524	.....	101.140	.....	24.980	.....
June	6.342	.....	92.058	.....	37.115	.....
July	6.802	.....	85.808	.....	40.000	.....
August	6.596	.....	80.115	.....	40.000	.....
September	6.542	.....	76.300	.....	40.000	.....
October	6.517	.....	72.385	.....	40.000	.....
November	6.679	.....	68.587	.....	40.000	.....
December	6.231	.....	66.115	.....	40.000	.....
Year	6.720	.....	87.351	.....	35.665	.....

(*a*) Antimony quotations in cents per pound, for ordinary brands. (*b*) Quicksilver in dollars per flask of 76 lb. (*c*) Platinum in dollars per ounce troy.

Pig Iron[1]

	Bessemer		Basic		No. 2 Foundry	
	1931	1932	1931	1932	1931	1932
January	17.50	16.00	17.00	15.00	17.00	15.50
February	17.29	15.68	16.79	14.68	16.79	15.18
March	17.00	.....	16.50	.....	16.50	.....
April	17.00	.....	16.50	.....	16.50	.....
May	17.00	.....	16.38	.....	16.44	.....
June	17.00	.....	15.50	.....	16.00	.....
July	17.00	.....	15.50	.....	16.00	.....
August	17.00	.....	15.50	.....	16.00	.....
September	17.00	.....	15.50	.....	16.00	.....
October	16.92	.....	15.42	.....	16.00	.....
November	16.50	.....	15.00	.....	16.00	.....
December	16.25	.....	15.00	.....	15.75	.....
Year	16.96	.....	15.88	.....	16.25	.....

Iron in dollars per long ton. [1]F.o.b. Mahoning and Shenango Valley furnaces; freight to Pittsburgh, $1.76.

Mining Share Prices—Month of February, 1932

Company Name	High	Low	Last	§Most Recent Dividend
NEW YORK STOCK EXCHANGE				
Alaska Juneau	16½	13	13¾	Ja. 9, Fe. 9, 1932 Q 0.12½
American Metal	5.25	4.25	4.75	December, 1930 Q 0.25
Am. Metal, pfd	17	15	16	Au. 22, Se. 1, 1931 Q 1.50
Am. Sm. & Ref	17⅜	12⅜	15½	Ja. 15, Fe. 1, 1932 Q 0.12½
Am. Sm. & Ref. 2a pfd.	51	48	48	Fe. 5, Mh. 1, 1932 Q 1.50
Am.Sm. & Ref. 1st pfd.	83	70	70	Fe. 5, Mh. 1, 1932 Q 1.75
Am. Zinc, L. & S.	3.00	2.62½	3.00	May, 1917 1.00
Am. Zinc, L. & S., pfd.	22⅜	19⅞	19⅞	October, 1930 Q 1.50½
Anaconda	11½	8.75	9.37½	Jy. 11, Au. 17, 1931 Q 0.37½
Andes Copper	3.87½	3.87½	3.87½	Ja. 10, Fe. 9, 1931 Q 0.25
Butte C. & Z.	1.12½	1.00	1.00	December, 1929 0.50
Butte & Superior	0.75	0.62½	0.75	December, 1929 Q 0.50
Callahan Zinc-Lead	0.50	0.37½	0.37½	December, 1920 0.50
Cal. & Hecla	3.25	3.00	3.00	June, 1930 0.50
Cerro de Pasco Copper	13⅜	9.75	11½	Ja. 14, Fe. 1, 1932 Q 0.25
Dome Mines	9.75	8.12½	9.62½	Mh. 31, Ap. 20, 1932 Q 0.25
Federal M. & S.	....	....	26*	June, 1927 10.00
Federal M. & S., pfd.	....	....	36*	My. 25, Je. 5, 1931 Q 1.75
Freeport Texas	19⅜	16½	18⅜	Fe. 15, Mh. 1, 1932 Q 0.50
Granby Con	7.25	6.00	6.00	Ja. 15, Fe. 1, 1932 Q 0.12½
Greene Cananea	....	....	15*	July, 1930 0.75
Homestake	122	110	115	Fe. 20, Fe. 25, 1932 MX 1.65
Howe Sound	14	10½	10⅝	De. 31, Ja. 15, 1932 Q 0.50
Inspiration	3.50	2.87½	3.00	July, 1930 0.50
International Nickel	8.87½	7.37½	7.50	De. 1 De. 31, 1931 Q 0.05
Int. Nickel, pfd.	82	79	79½	Ja. 2 Fe. 1, 1932 Q 1.75
Kennecott Copper	11⅜	8.87½	9.25	De. 17, Ja. 2, 1932 Q 0.12½
McIntyre Porcupine	15⅞	14⅜	14⅜	Fe. 1, Mh. 1, 1932 Q 0.25
Magma Copper	7.37½	6.87½	7.00	De. 31, Ja. 15, 1932 Q 0.12½
Miami Copper	3.50	3.00	3.12½	August, 1930 0.375
Mother Lode	0.37½	0.25	0.37½	June, 1930 0.10
National Lead	90	86	90	Mh. 18, Mh. 31, 1932 Q 1.25
National Lead, pfd. A.	121	116	121	Ja. 15, Fe. 1, 1932 Q 0.62½
National Lead, pfd. B.	100	100	100	Ap. 22, My. 2, 1932 Q 1.50
Nevada Con	5.62½	4.37½	4.37½	De. 17, De. 31, 1931 Q 0.10
Noranda Mines	14⅜	13½	14	De. 15, De. 29, 1931 0.50
Park Utah Con	1.25	1.00	1.00	July, 1929 Q 0.20
Patino M. & E.	9.00	6.87½	7.75	December, 1929 4s.
Phelps Dodge	7.25	6.00	6.00	Je. 18, Jy. 1, 1931 Q 0.25
Rand Mines	....	....	20*	August, 1930 1.52
St. Joseph Lead	10⅜	8.37½	9.37½	Mh. 10, Mh. 21, 1932 Q 0.15
Seneca Copper	0.37½	0.25	0.37½	....
Tennessee Corp	2.50	2.00	2.00	Fe. 28, Mh. 16, 1931 Q 0.125
Texas Gulf Sulphur	25⅝	22½	24⅜	Mh. 1, Mh. 15, 1932 Q 0.50
U. S. S. R. & M.	16⅞	15⅛	16	De. 31, Ja. 15, 1932 Q 0.25
U. S. S. R. & M., pfd.	38½	38	38	De. 31, Ja. 15, 1932 Q 0.87½
Utah Copper	59	40	40	De. 17, De. 31, 1931 Q 1.00
Vanadium Corp.	17½	12½	16¾	Fe. 2, Fe. 16, 1931 Q 0.75
NEW YORK CURB EXCHANGE				
Aluminum Co. of Am.	57½	46½	48¾	....
Alum. Co. of Am., pfd.	63	57	57	Ap. 1, 1932 Q 0.75
Anglo-Chilean Con	0.43¾	0.37½	0.37½	4 shares Cosach B June, 1931
Bunker H. & S.	24	22½	23½	De. 31, Ja. 5, 1932 Q 1.50
Carnegie Metals	1.62½	1.00	1.50	....
Comstock Tunnel	0.43¾	0.37½	0.37½	....
Con. Coppermines	1.00	0.87½	0.87½	....
Copper Range	....	....	2.00*	July, 1930 0.25
Cresson Consolidated	0.25	0.18¾	0.25	Je. 30, Jy. 10, 1931 Q 0.01
Cusi Mexicana	0.31¼	0.25	0.31¼	....
Evans-Wallower	0.37½	0.25	0.37½	....
Federated Metals	6.00	5.50	5.50	Ja. 5, Ja. 15, 1931 Q 0.25
Hecla Mining	4.75	4.00	4.00	Fe. 15, Mh. 15, 1932 Q 0.10
Hollinger Consolidated	4.50	4.25	4.37½	Fe. 11, Fe. 25, 1932 M 0.05
Hudson Bay M. & S.	2.25	1.75	1.87½	....
Iron Cap Copper	0.62½	0.62½	0.62½	....
Mayflower Associates	26½	24½	24½	Mh. 1, Mh. 15, 1932 Q 0.90
Mining Corporation	1.12½	1.00	1.00	December, 1929 0.12½
New Jersey Zinc	27½	25½	26	Ja. 20, Fe. 10, 1932 Q 0.50
Newmont Mining	12⅞	10	11½	Mh. 31, Ap. 15, 1931 Q 1.00
N. Y. & Honduras Ros.	13⅜	13½	13⅜	Ja. 19, Ja. 30, 1932 X 0.37½
Nipissing	0.75	0.75	0.75	Mh. 31, Ap. 20, 1931 Q 0.075
Ohio Copper	0.12½	0.06½	0.12½	September, 1926 0.03
Premier Gold	0.56½	0.50	0.50	De. 14, Ja. 4, 1932 Q 0.03
Quincy	2.00	2.00	2.00	March, 1920 1.00
Shattuck Denn	2.00	1.75	1.75	....
So. Am. Gold & Plat.	....	....	0.50*	....
Teck-Hughes	4.00	3.62½	3.62½	Ja. 16, Fe. 1, 1932 Q 0.15
Tonopah Mining	....	....	0.31¼*	October, 1929 0.07½
United Eastern	....	....	0.06¼*	....
United Verde Ex	3.87½	3.25	3.75	Ja. 2, Fe. 1, 1932 Q 0.25
Utah-Apex	0.5	0.50	0.50	Decem er, 1929 0.02
Yukon Gold	....	....	0.12½*	June, 1918 0.02
BOSTON STOCK EXCHANGE				
Isle Royale	1.75	1.62½	1.75	March, 1930 0.50
Mohawk Mining	18⅜	16	18	Ja. 30, Mh. 1, 1932 Q 0.25
Old Dominion	0.50	0.50	0.50	December, 1918 1.00
Utah Met. & Tunnel	0.45	0.35	0.36	December, 1927 0.30
¶MONTREAL				
Asbestos Corp	0.30	0.25	0.25	January, 1926 1.50
Asbestos Corp., pfd.	1.00	0.65	0.65	January, 1930 1.75
Consolidated M. & S.	0.72	0.64¾	0.69	De. 23, Ja. 15, 1932. SAX 1.25

Company Name	High	Low	Last	§Most Recent Dividend
SPOKANE				
Constitution	....	....	0.06¾*	....
Dayrock	....	....	0.95*	March, 1930 0.03
Golconda	0.20	0.18¾	0.20	July, 1929 0.02
Jack Waite Con	0.22	0.19	0.19	....
Los Lugos	....	....	0.01¾*	....
Mexican Premier	0.07½	0.06½	0.06½	January, 1930 0.01
Montana Mines	....	....	0.09*	....
Noble Five	....	....	0.03*	January, 1930 0.005
Pend Oreille	1.03	0.90	0.97	....
Sherman Lead	0.27	0.27	0.27	March, 1930 0.01
Sidney	....	....	0.05*	October, 1929 0.01
Sunshine Mining	1.18	1.10	1.12	De. 10, De. 23, 1931 X 0.02
Tamarack & Custer	0.15¼	0.15¼	0.15½	September, 1924 0.25
SALT LAKE CITY				
Chief Con	34	31	31	February, 1928 0.10
Combined Metals	5.50	4.75	4.75	....
Eureka Bullion	.21	17½	20½	....
Eureka Lilly	10½	9.00	9.00	....
Moscow Silver	60	35	60	....
New Quincy	4.00	2.25	2.25	September, 1929 Q 0.10
North Lily	69	62	65	January, 1930 0.15
Park Bingham	....	....	1.75*	....
Park City Con	10½	9.25	9.25	....
Rico Argentine	14½	14½	14½	December, 1929 0.03
Silver King Coalition	3.47½	2.85	3.00	De. 20, Ja. 2, 1931 Q 0.15
Tintic Lead	95	80	90	....
Tintic Standard	2.80	2.45	2.45	De. 10, De. 24, 1931 Q 0.10
Walker Mining	75	57	57	April, 1930 0.075
¶ TORONTO				
Abana Mines	0.05¾	0.04½	0.04½	....
Amulet Mines	0.19	0.16	0.16	....
Barry-Hollinger	0.90	0.80	0.80	....
Base Metals	0.08	0.06	0.06½	....
Canam Metals	0.03	0.03	0.03	....
Castle-Trethewey	0.14	0.13	0.13	....
Central Manitoba	0.08*	0.06	0.07	....
Falconbridge	1.00	0.80	0.90	....
Granada Gold	1.11	0.90	1.00	....
Howey Mines	0.36	0.30	0.30½	....
Keeley	0.23	0.23	0.23	March, 1928 0.04
Kirkland Lake	0.51	0.42	0.42	....
Lake Shore	29.10	28.00	28.00	Mh. 1, Mh. 15, 1932 Q 0.50
Sherritt Gordon	0.64	0.49	0.50½	....
Siscoe Gold	0.62½	0.54	0.60	Mh. 21, Mh. 31, 1932 I 0.04
Sylvanite	0.65	0.52½	0.56	De. 5, De. 31, 1931 SA 0.02
Treadwell Yukon	1.30	1.01	1.25	....
Ventures	0.52	0.43	0.45	....
Vipond Con	0.40	0.26	0.35	April, 1927 0.03
Waite-A.-M	0.92	0.75	0.80	....
Wright-Hargreaves	2.87	2.55	2.60	Mh. 10, Ap. 1, 1932 Q 0.02½
ST. LOUIS, CINCINNATI, AND DENVER				
Consolidated L. & Z.	1.00	1.00	1.00	December, 1929 0.125
Eagle-Picher	5.12½	4.75	4.75	April, 1930 0.20
Eagle-Picher, pfd	....	....	89*	De. 31, Ja. 15, 1931 Q 1.50
Empire Chief	0.00¾	0.00¾	0.00¾	....
Star of Utah	0.12	0.11	0.12	....
Tungsten Prod	0.09	0.05	0.05	....
Ute Gold	0.10	0.06	0.10	....
SAN FRANCISCO, LOS ANGELES, AND COLORADO SPRINGS				
California Copper	0.25	0.12½	0.25	....
(a)Golden Cycle	12	10	10	Fe. 29, Mh. 10, 1932 Q 0.40
(a)Granite Gold	0.18	0.15	0.15	De. 23, Ja. 2, 1932 Q 0.01
Idaho-Maryland	1.90	1.47½	1.90	....
Tom Reed	0.28	0.20	0.26	De. 15, Ja. 1, 1932 Q 0.03
(a) United Gold	0.05	0.03½	0.03½	Oct. 15, No. 1, 1931 0.01

‡LONDON

Company Name	High	Low	Last	Most Recent Dividend (Per Cent, Unless Noted)
Anglo American	7/7½	6/3	6/3	22½, Feb. 19. 1930
Aramayo Mines	16/3	15/3	16/3	5, Oct. 23, 1929
Burma Corporation	9/7½	8/4½	9/4½	1¾, Dec. 10, 1930(b)
Camp Bird	–/4	–/3	–/3	16¾, April 25, 1928
Fresnillo	9/–	9/–	9/–	25 cents, Feb. 12, 1930
Frontino & Bolivia	16/6	15/7½	16/3	2½, July 1, 1931
Lake View & Star	9/1½	7/3	8/10½	11½, 1921
Mexican Corporation	3/7½	3/–	3/6	10, Oct. 24, 1929
Oroville Dredging	3/9	1/10½	3/9	37½, May 16, 1930
Rhodes.-Select. Trust	6/7½	5/6	5/9	....
Rio Tinto	18⅝	14½	17½	10s, May 1, 1931
St. John del Rey	18/7½	17/9	18/–	9d, No. 20, 1931
San Francisco Mines	8/10½	7/1½	8/3	10, Jan. 7, 1931
Santa Gertrudis	6/7½	6/–	6/3	9d., July 9, 1931
Selukwe	1/10½	1/3	1/9	6¾, April 2, 1917
So. American Copper	–/9	–/6	–/6	....
Tanganyika	22/4½	17/6	20/7½	2½, Aug. 2, 1930
Wiluna Gold	12/10½	8/1½	12/6	....
Union Minière	4,515	3,615	4,230	240 francs, July, 1931

Salt Lake Stock Exchange courtesy J. A. Hogle & Company. ¶These figures are quoted in Canadian dollars. *Not traded in. †Belgian francs. ‡The above London quotations are for the month ended Feb. 23. Q, quarterly; I, initial; M, monthly; X, extra; SA, semi-annual; L, liquidating B, plus bonus. §Total dividends paid during the month will be found tabulated in the Market and Financial News Section. Dates here given are those of closing of books and payment of dividend. (a) Bid prices. (b) 1½% (2 annas) dividend, and bonus of ½% (1 anna). (c) Closing bid prices. F.W., four weeks.